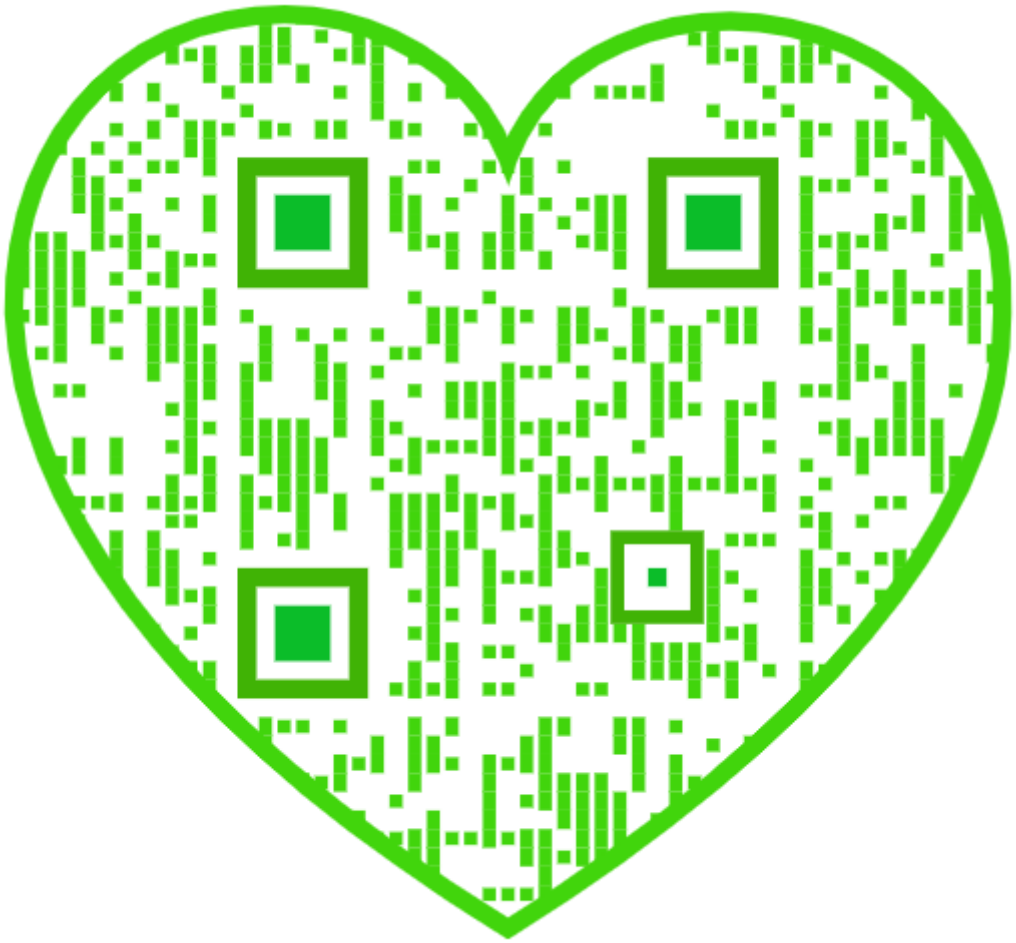

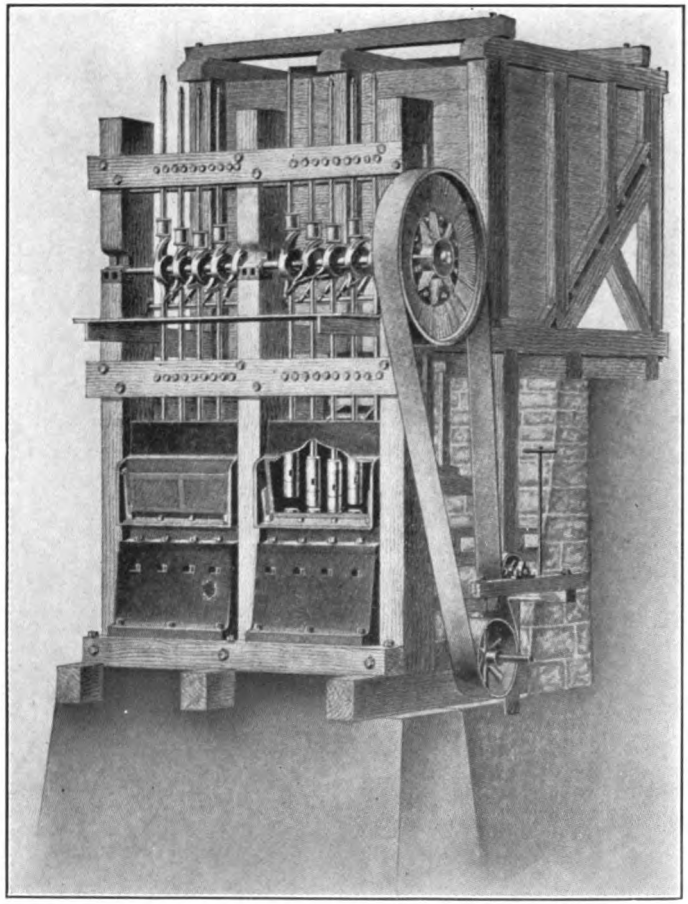

Frontispiece

FIG. 1. 10-Stamp Mill with Cast-iron Anvil Blocks.

STAMP MILLING

AND

CYANIDING

BY

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To the memory
OF
J. A. T. AND J. K. R.
HIS FATHERS IN ENGINEERING,
THE AUTHOR DEDICATES THIS WORK

PREFACE

This book is designed for use as a text. It is, therefore, properly and of necessity a compilation. The subject matter and method of presentation have been developed from my lectures on the metallurgy of gold and silver, and will, I trust, meet with the approval of other students of the metallurgic art.

While every effort has been made to avoid errors, it is scarcely to be hoped that none will be found, and I shall appreciate having my attention called to such as are discovered.

I have levied freely upon all available sources of information. Especially am I beholden to the files of the *Mining (and Scientific) Press*, the *Engineering and Mining Journal*, the *Mining and Engineering World*, the *Mining Magazine* (of London), the Transactions of the American Institute of Mining Engineers and of the Institution of Mining and Metallurgy. I have also made use of all available texts, and lastly have drawn in many cases from my own ten years of professional practice.

In every case I have tried to acknowledge the source from which information has been taken, but doubtless I have failed to do so in certain instances; for even as "we are part of all whom we have met," so our knowledge is part of all that we have read, and after many years of reading full acknowledgment is no longer a possibility.

For aid in illustrating this book my thanks are due to the following manufacturers: Allis-Chalmers Co., Milwaukee; Blaisdell Co., Los Angeles; Braun-Knecht-Heimann Co., San Francisco; Chalmers and Williams, Chicago; Chrome Steel Works, Brooklyn; Colorado Iron Works, Denver Engineering Works, and Dorr Cyanide Machinery Co., Denver; Fairbanks-Morse and Co., Chicago; Hammond Mfg. Co., Portland, Ore.; Hardinge Conical Mill Co., New York; Joshua Hendy Iron Works, San Francisco; Mine and Smelter Supply Co., Denver; Monarch Engineering Co., Baltimore; Moore Filter Co., New York; Oliver Continuous Filter Co., and Pacific Pipe and Tank Co., San Francisco; Power

and Mining Machinery Co., Cudahy, Wis.; Stearns-Roger Mfg. Co., Denver, and L. C. Trent Engineering Co., Los Angeles; also to the publishers of the *Engineering and Mining Journal*, and to the secretary of the American Institute of Mining Engineers.

I desire to acknowledge aid received from the following mining engineers and metallurgists: Messrs. W. B. Chidester, Edgar A. Collins, David Goodale, Henry Hanson, J. W. Hutchinson, Merrill Metallurgical Co., A. R. Parsons, Lloyd Robey, Harry C. Robinson, and R. Wheeler.

For faithful and invaluable help in typing the manuscript I am indebted to Miss Helene T. Bruckman; and for aid in proof-reading the bibliography, to my assistant Mr. Henry E. Doelle.

To Mr. Frank A. Ross I am particularly obligated for valuable suggestions and for reading the typescript.

To my colleague Prof. Hallet R. Robbins, especial thanks are due for critical scrutiny of the final proof.

FRANCIS A. THOMSON.

STATE COLLEGE OF WASHINGTON,
February, 1915.

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PART I

MILLING AND AMALGAMATION

STAMP MILLING AND CYANIDING

CHAPTER I

GOLD AND SILVER

The modern metallurgy of gold, taking the term in its purely commercial sense, and excluding from consideration the recovery of the precious metals as a by-product of smelting, concerns itself with the three processes, amalgamation, cyanidation, and — to a continually decreasing extent — chlorination.

The metallurgy of silver, which for many years has lagged far behind that of its sister metals, is now making rapid strides and, while at the present time amalgamation still plays an important part, it seems destined to yield almost entirely to cyanidation.

GOLD

Physical and Chemical Properties. Gold is the one metal that is yellow in color, malleable, and of high specific gravity (19.6); these properties serve at once to distinguish it from all others. Gold melts at about 1060° C. and is sensibly volatile above 1200°. It has high electrical conductivity (76.7), crystallizes in the isometric system (usually in octahedra), alloys readily with nearly all other metals, and has at least one allotropic form, and possibly two. Chemically, gold forms compounds with the halogens, oxygen, sulphur, and silica, also with the salts of thiosulphuric and hydrocyanic acids.

Occurrence. In nature only two forms are positively recognized. Native, or free, gold is by far the most common occurrence. Less frequently it is found associated with tellurium in various combinations. Selenide of gold is said to have been discovered but its existence is still a matter of doubt.

Metallurgical Properties. From the metallurgist's point of view five properties of gold are of prime importance. These are:

1. High specific gravity.
2. Formation of alloys with mercury.
3. Formation of alloys with lead, copper, and copper sulphide.
4. Solubility in solutions of cyanide salts.
5. Formation of a soluble chloride by contact with chlorine gas.

The first property is purely physical. It finds an important application in the recovery of gold in riffles, on blankets, etc., in connection with placer mining and is a distinct aid in the milling of gold ores.

The second property — the formation of an alloy, or so-called amalgam, with mercury — is the basis of the amalgamation process, and was for many years the most important factor in gold metallurgy.

The third property is utilized in lead and copper smelting, and while extremely important in point of production, will not be further discussed in this work.

The fourth property — the solubility of gold in salts of hydrocyanic acid — is the fact upon which the cyanide process is based; known, but unused, twenty-five years ago, this chemical fact has revolutionized the art of gold recovery.

The fifth property — the formation of a soluble gold chloride by means of the action of chlorine gas upon moist gold — was for many years second only to amalgamation as a means of gold recovery. In spite of many ardent advocates, however, the process has been steadily losing ground.*

SILVER

Physical and Chemical Properties. Silver, next to platinum and gold, is the most valuable of the more common metals. It is distinguished by its white color, brilliant luster, malleability, and hardness. In electrical conductivity silver excels all other metals. It melts at 962° C. and is readily volatile at higher temperatures. Its specific gravity (10.5) is about half that of gold.

Chemically, silver is much more active than gold. Nitric acid attacks it readily, and it is soluble in hot concentrated sulphuric

* One of the most interesting controversies in metallurgical literature is to be found in the Eng. and Min. Jour., Vols. 78 and 79, where Phillip Argall and W. E. Greenwalt debate at great length the comparative merits of cyanidation and chlorination.

acid. Owing to the insolubility of the chloride, hydrochloric acid attacks it only superficially. Many salts of silver are soluble in water and this fact has led to numerous proposals for extracting it from its ores.

Occurrence. Silver occurs in nature in a greater variety of forms than any other well known metal.

Native silver in threads and crystals is frequently found either alone or associated with gold, or with native copper. Native silver-amalgam is not unknown. Argentite Ag_2S containing 86 per cent silver is one of the commonest silver minerals. The sulph-antimonides and sulpharsenides of silver are next in importance; among the most usual are the light and dark ruby silvers, proustite Ag_3AsS_3 and pyrargyrite Ag_3SbS_3, also stephanite $AgSbS_4$. The haloid silver minerals, especially cerargyrite the chloride, and embolite the chlorobromide, are important in the upper, oxidized portions of most silver veins.

Nearly all copper minerals, and practically all lead minerals, carry more or less silver; such ores, however, are treated almost altogether by smelting.

Metallurgical Properties. Probably more processes have been proposed for the extraction of silver from its ores than for any other metal. To mention all of the properties upon which these have been based would be useless.

The most important are as follows:

1. The property possessed by silver and its halogen compounds of forming an alloy or amalgam with mercury.
2. The property possessed by copper, copper sulphides and metallic lead of dissolving or alloying with silver in various forms.
3. The solubility of silver chloride in a solution of sodium hyposulphite.
4. The solubility of nearly all silver minerals in cyanide solutions.

The first-mentioned property has, even more than is the case with gold, been responsible in the past for a large part of the world's supply of silver. Aside from the smelting methods mentioned under the second heading, it is safe to say that amalgamation, until the last few years, has supplied over 90 per cent of the silver production of the world.

For some years hyposulphite leaching was in vogue but wet processes were crude then and mechanical difficulties, coupled in

many cases with the necessity of a preliminary roast with salt,— to convert other silver compounds into chlorides,— gradually led to its abandonment. The mechanical perfection of the present-day cyanide plants would solve many of the problems of hyposulphite lixiviation, but it is doubtful if the latter has any advantages over cyaniding, and its revival does not seem likely.

Cyaniding of silver ores is a new field. Ten years ago it was scarcely heard of. To-day Mexico and Nevada are producing millions annually by this means.

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CHAPTER II

PRINCIPLES OF AMALGAMATION

Amalgams. An amalgam is an alloy of mercury with other metals. The word amalgam is derived from the greek '*malagma*'— poultice — referring, of course, to the pasty nature of mercury alloys.

On account of the low fusing point of mercury −42° F., its alloys with other metals were early investigated, but so far as we are aware no commercial uses were made of them for many years after their existence was known. Tin and bismuth amalgams are used for silvering mirrors and silver-tin amalgams with small additions of other metals are much used in dental work. The amalgams of gold and silver, however, are those most important for the metallurgist.

While much remains to be learned concerning the exact constitution of these precious-metal amalgams, it may be safely said that there are certain definite chemical compounds of both gold and silver with mercury. These compounds, in turn, form solid solutions with an excess of mercury, so that, while gold and silver are only to an extremely limited degree soluble in mercury, their amalgams will remain in suspension in it to almost any extent.

Recent investigations into the constitution of gold and silver amalgams by T. T. Read * have demonstrated the following interesting facts:

1. That there are two definite compounds of gold and mercury, one carrying 13.62 atomic per cent of mercury, corresponding nearly to the formula $Au_{19}Hg_3$, and the other 17.44 atomic per cent, corresponding nearly to $Au_{19}Hg_4$.

2. That gold is soluble in mercury at 20° C. to the extent of about 0.06 per cent; that this solubility increases to about 0.25 per cent at 100° C. and probably decreases to zero at the freezing point of mercury.

* Trans. A.I.M.E., Vol. 37, page 64 *et seq.*

3. That while no definite results have been obtained with silver it is probably the case that compounds of silver and mercury exist similar to those mentioned for gold.

Amalgamation is, of course, the art of forming or making amalgams, but in metallurgical parlance, the word is limited to the means adopted for the recovery of gold and silver from its ores with the aid of mercury. The use of mercury for this purpose is one of the ancient arts. Vitruvius and Pliny both mention its employment for the extraction of gold from gold cloth. As it had long been the custom to recover gold from both lode and placer mines with the aid of hides or long-napped cloth, it would be but a short step to connect these two ideas and make the mercury complete the extraction of gold from the hide or blanket concentrates. This knowledge of amalgamation as applied to gold ores was so well nigh lost sight of in the middle ages, that it reappears in the sixteenth century as a secret art.

For the application of the amalgamation process to the recovery of silver, we are indebted to Bartolome de Medina, a Mexican miner, who, at Pachuca, in 1557, invented the 'patio process.' This was the first and, for many hundreds of years, the most important silver amalgamation method. The fact that silver usually occurs not native, but in combination with other elements, explains why amalgamation was not applied to silver ores until long after gold amalgamation was an established art. The ignorant Mexican miner solved, probably by accident, the problem that had baffled the European metallurgists.

In order that amalgamation or union between gold (or silver) and mercury can take place, they must be brought into intimate contact. Presuming that by previous treatment the particles of gold have been freed from adhering gangue-matter, there are three methods by which the desired contact may be secured:

1. *By grinding-contact.*
2. *By surface-contact.*
3. *By immersion-contact.*

(1) The first method, grinding-contact, is the oldest, and if we lay aside the commercial considerations of time and mercury consumed, it is probably the most efficient. It is still practiced exclusively for the amalgamation of silver ores. The grinding in contact with mercury is accomplished in iron or copper pans, or in areas enclosed by masonry, all of which lend themselves conven-

iently to the addition of chemicals and in most cases to the use of hot water.

(2) The surface-contact method of amalgamation did not appear until the happy idea of the California millmen, in 1860, led them to coat with mercury the surface of sloping copper tables, over which the crushed ore and water (pulp) was allowed to flow in a thin film. T. K. Rose suggests that this idea came from the knowledge gained in silver amalgamation with copper pans, where it had been found that the amalgam formed crusts on the sides of the copper vessel. In any event the introduction of copper amalgamating-plates revolutionized the milling of gold ores.

The action of the copper plate coated with quicksilver is extremely simple. As the sheet or film of pulp flows over the plate the heavier particles of gold naturally seek the bottom of the film and come in contact with the mercury surface. The mercury at once seizes upon the gold particles and causes them to adhere to the plate. If, as has been pointed out by Read (*loc. cit.*), the particle of gold is smaller in diameter than the thickness of the mercury film, no disturbing effect is produced; if the diameter of the particle be greater than the thickness of the film, then the surface tension of the mercury causes it to pull the gold particle down to the plate with considerable force, which of course is assisted by the attraction of gravity upon the gold particle itself. One of the most important points in favor of plate amalgamation is, that amalgam is a much more efficient gold catcher than barren mercury, and with the thin film existing on the copper plate, the gold is always exposed to the attraction of amalgam when once the operation has been started. Another feature of value in connection with plates is that they are readily accessible for inspection or for removal of accumulated amalgam. This last feature has at times, of course, been provocative of theft.

(3) Probably no generally discredited method has been the subject of more patents and inventions than the scheme of forcing previously crushed ore to immerse itself for a brief moment in a mercury bath, in the belief that by so doing, the gold would be taken up by the mercury in the form of amalgam.

These appliances are not in use in standard mills in the United States or Canada, and while European and Australian practice favored them for many years, they are gradually giving way to copper plates, or to other, usually less efficient, surface-contact

amalgamators. The reasons why immersion-amalgamators are less successful than plates may be summarized as follows:

(*a*) It is practically impossible to get any such path of contact as is afforded by a long copper plate. (*b*) With an ore carrying sulphides or other heavy minerals, these collect on the surface of the bath and complicate the operation. (*c*) The time of contact is so short and the likelihood of the gold particles being surrounded with a film of air or of other ore particles is so great, that the gold is not readily 'wetted' by the mercury and hence amalgamation does not take place. (*d*) Amalgam is a more efficient gold-saver than mercury, and in a mercury bath the tendency for the gold amalgam to settle away from the balance of the quicksilver is sufficient to make the contact mainly one of ore and mercury, instead of one of ore and amalgam as in plate work. (*e*) Large amounts of mercury are kept tied up in the machine and a large loss due to mechanical subdivision of mercury into small particles is inevitable.

Comparing and summarizing we find:

1. *That grinding-contact is efficient but expensive and consumes much mercury. As a final amalgamation treatment it has been abandoned for all but silver ores and rich gold concentrates.*

2. *That surface-contact with copper plates is efficient and inexpensive and is almost universally employed for gold ores.*

3. *That immersion-contact has little to recommend it and has practically been abandoned.*

Flouring and Sickening. Reference has been made to loss of mercury by its being broken up into small globules which will not recoalesce; this is known technically as 'flouring.' 'Sickening' of mercury is a term closely connected with flouring and by many writers not clearly distinguished from it. T. A. Rickard * defines flouring "as the minute subdivision of mercury by mechanical causes, and sickening as the rendering of such a condition permanent, by the intervention of a coating of some foreign substance, effectually preventing the coalescence or reunion of the globules previously formed." This illuminating distinction has been generally accepted and will be adhered to throughout this work.

* Stamp Milling of Gold Ores, 1st Ed., page 219.

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CHAPTER III

THE STAMP MILL AND ITS ACCESSORIES

The gravity stamp mill is at once the most primitive and the crudest crushing machine in existence, while at the same time the most satisfactory all-round milling-and-amalgamating device yet contrived. In its modern form the stamp mill (Fig. 1) consists essentially of a series of hammers (the stamps) falling upon a mixture of ore and water, inside a cast-iron box (the mortar). The resulting crushed ore (the pulp) is splashed out through a screen or grating in the end or side of the box.

Originating in Saxony, and first used in the United States in Virginia and Georgia, the stamp mill, in a primitive form, found its way to California and then to Gilpin County, Colorado. Here, free from the trammels of tradition, it speedily developed from an affair of wood, shod with iron and operated with the crudest gearing, to substantially its present form. The difference in conditions led, however, to a development along vastly different lines. The easily treated ores of California led the millmen to design their plants for the greatest crushing efficiency since amalgamation was a comparatively simple matter. In Gilpin County, however, the situation was reversed and crushing capacity had to be made decidedly secondary to the successful amalgamation of a comparatively refractory ore. From the California practice, since it has been found the more generally applicable, have grown the modern methods of stamp milling as employed in practically all parts of the world, with the exception of Australia where development has proceeded along somewhat independent lines.

The important parts of the stamp mill may conveniently be discussed under the following heads.

(1) *The stamp and its mechanism.*
(2) *The mortar.*
(3) *The foundation and framework.*
(4) *Accessory apparatus and operation.*

(1) THE STAMP AND ITS MECHANISM

The stamp proper constitutes the hammer which falls upon the particles of ore on the die or anvil, thus breaking the ore into fragments. It consists of the following parts as lettered in Fig. 2:

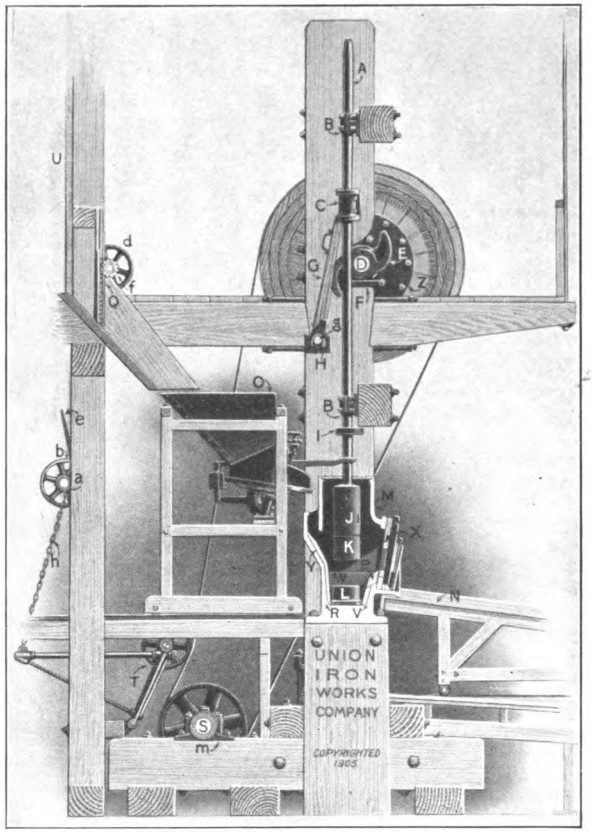

FIG. 2. Sectional Elevation of California Stamp Mill.

A. **The Stem** is the central shaft or spindle upon which are mounted the tappet, boss, and shoe. The stem is usually of soft steel from 3 to 4 in. in diameter and from 9 ft. 6 in. to 14 ft. 6 in. in length. It is cylindrical throughout, except the last 4 to 6 in. at each end, which are tapered to a diameter from $\frac{1}{4}$ to $\frac{1}{2}$ in. less than the diameter of the stem proper.

J. **The Boss, Boss-Head, or Head** (Fig. 2*J*), as it is variously called, is the cylindrical connection between the stem and the shoe.

It is of the same diameter as the latter and from 15 to 30 in. long. The upper end has a centrally located, accurately drilled, conical hole for the reception of the stem. At the lower end of the boss is a similar conical opening, larger and less accurately made, for the accommodation of the shoe-shank. At the small end of each of the conical openings is a rectangular hole at right angles to the axis of the cone, through which a key, or drift, may be driven to loosen either stem or shoe. Cast-iron bosses are commonly employed, but the tendency is toward the use of some higher grade material, such as chrome or manganese-steel.

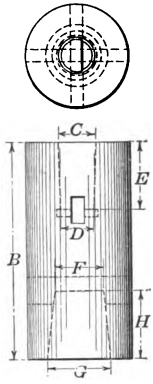

FIG. 2*J*. The Boss.

K. **The Shoe** (Fig. 2*K*) consists of a cylinder surmounted by a truncated cone or shank. The shoe is usually from 8 to 10 in. in diameter and from 6 to 10 in. high, exclusive of the shank. The shank is usually 5 to 6 in. high and tapers from a diameter of $4\frac{1}{2}$ or 5 in. at the bottom to $3\frac{1}{2}$ or 4 in. at the top. The shank is secured in the boss by means of wooden wedges ranged around it. These wedges are preferably made of medium-hard wood and should be so cut as to allow a space of about $\frac{1}{2}$ in. between the top of the shoe and the bottom of the boss, when the former is driven up tight.

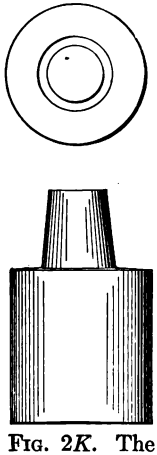
FIG. 2*K*. The Shoe.

Shoes are made of cast-iron, chrome-steel, manganese-steel, and other special steel alloys. The choice of material is a matter of the hardness of the ore and the facilities of the market. Where a mill has its own foundry and can use scrap-iron from about the plant, home made cast-iron shoes and dies are frequently economical. Where transportation charges are high, then the very best material, regardless of first cost, is advisable and chrome or other steel should be employed for both shoes and dies.

C. **The Tappet** serves to transmit the lifting power of the cam to the stamp. In principle it is simply a collar secured to the stem by a gib and keys. Formerly tappets were secured to the stem by being screwed upon an upset thread.

This is scarcely, if ever, seen now-a-days. The detailed construction of a two-keyed tappet is shown in Fig. 2*C*, where *S* is the stem, *G* is the gib, and *KK* the keys. The projecting portions at the top and bottom of the tappet are the surfaces against which the cam operates; when one end is worn the tappet is turned upside down. As in the case of the boss, tappets are frequently of cast iron, but the tendency is toward the use of some one of the special steels already mentioned.

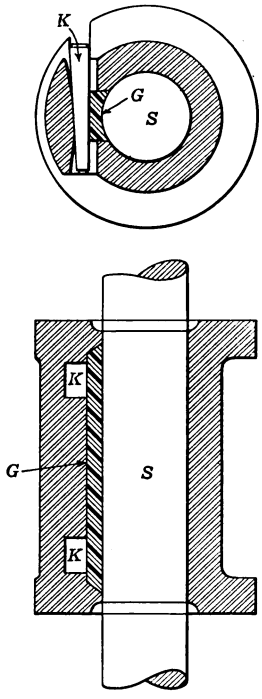

FIG. 2*C*. Two-key Tappet.

L. **The Die** while not properly a part of the stamp may be conveniently discussed under this heading. It is a cylindrical anvil, slightly larger in diameter than the shoe, and is usually provided with a square or octagonal base to secure it from movement when placed in the mortar. Dies may advantageously be made of somewhat softer material than the shoes, owing to the fact that the ore lying upon them protects them from the blow. As pointed out in discussing shoes, wherever freight is a serious item the dies should be of the best material available.

Table I, taken from the catalogue of the Stearns-Roger Mfg. Co., Denver, Colo., shows the usual variations in weights and dimensions of stamp parts constructed by them, where the complete stamp varies from 850 to 1600 lb. in weight.

The stamp is lifted by means of a double-armed cam from which it drops when the full lift is reached. Single- and triple-armed cams have been used but are no longer seen; the Saxon and Cornish stamp mills used a shaft of large diameter with projecting teeth for cams. These may still be seen in the countries of their origin. The important features of the lifting mechanism are the cam, the cam-shaft, and the pulley, or 'bull-wheel' as it is often called.

The Cam (*E*, Fig. 2, page 14) consists of a circular hub, from which project two arms. The face of each arm is curved to an

TABLE I

Dropping weight of stamp in lb.	850	950	1050	1250	1500	1600
Stem, diam.....in.	3½	3½	3½	3¾	4	4
Stem, length.....ft.	14	14	14	14½	15½	16
Stem, weight.....lb.	365	395	458	545	661	683
Tappet.....lb.	135	135	156	178	212	212
Bosshead.....lb.	225	252	270	342	420	498
Shoe, weight.....lb.	145	170	170	185	207	207
Shoe, dimensions, diam. × height.....in.	8½×8	9×8	9×8	9×9	9×10	9×10
Bosshead, dimensions.....in.	8½×18	9×17½	9×19	9½×22	9½×26½	9½×30½
Die, dimensions, diam. × height.....in.	8½×6	9½×6	9½×7	9½×7	9½×8	9½×8
Die, weight.....lb.	106	115	140	140	160	160
Cam shaft (10 stamps), dimensions in. diam. × ft. length...	5⅝×14½	5½×14½	6×14½	6⅛×14½	6⅛×15	6 ⅛ × 15
Cam shaft, weight.....lb.	1115	1170	1400	1740	1800	1800
Cam shaft pulley (10 stamps), dia. × face.....in.	72×13	78×15	84×17	84×19	84×22	84×22
Cam, 5 in. to 7 in. drop, diam. × weight.....	26×155	26×155	27×180	28×210	28×220	28×220
Cam, 6 in. to 8 in. drop, diam. × weight.....	28×165	28×165	29×192	30×230	30×242	30×242
Cam, 7 in. to 9 in. drop, diam. × weight.....	30×175	30×175	31×205	32×248	32×263	32×263
Cam, 8 in. to 10 in. drop, diam. × weight.....	32×185	32×185	33×220	34×260	34×278	34×278

involute of a circle, the radius of which is the distance from center to center of cam-shaft and stem. The exact shape of this cam-curve is a matter of extreme importance. Much of the noise, vibration, and breakage in poorly designed mills is due to errors in the design of the cams or in the setting of the shaft. The cam-curve must be so constructed as to fulfill the following conditions:

First. The engaging surfaces of cam and tappet must remain parallel throughout the entire lift to insure the minimum of jar and friction and to avoid excessive thrust from the cam. Fig. 3 illustrates this point.*

Second. The velocity of lift must be constant and the amount of lift must be directly proportional to the arc traversed by the cam-shaft.

Third. For two-armed cams this arc must be considerably less than 180°.

The normal involute mentioned fulfills the first two conditions.† This normal involute is the curve generated by the end of a tight string as it is unwound from a circle. The string is always tangent to the circle, consequently the curve it generates must be normal to the tangent at each point. It is also evident that the

* Mark R. Lamb, Eng. & Min. Jour., Vol. 88, page 66.

† For complete discussion see Henry Louis' "Handbook of Gold Milling," 2nd Ed., pages 200–206 and 557–569.

extension of the string is equal to the length of the arc from which it has been unwound. Considering Fig. 3 in which the cam-curve is the normal involute of a circle whose radius is $A + B + C$, where A is the shaft radius, B the clearance between shaft

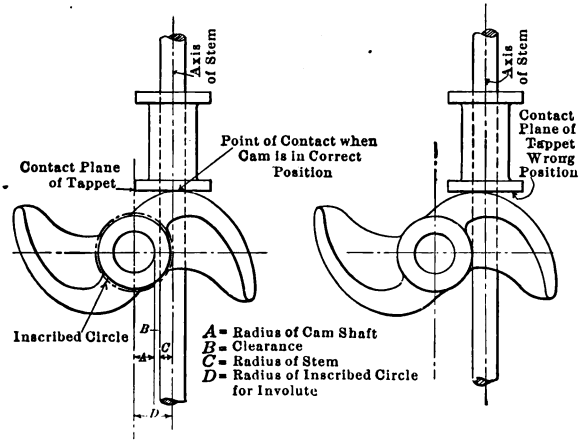

FIG. 3. Position of Cam.

and stem, and C the radius of the stem; we find the axis of the stem tangent to the inscribing circle; and since the curve of the involute is normal to the tangent of its inscribing circle, the cam-curve at the point of contact is normal to the stem axis and parallel to the tappet face. This fulfills the first condition.

We have said that the string in generating the involute increases by the length of arc unwound, *i.e.*, the point of the string has moved along the tangent and away from the center of the circle, a distance equal to the rectified arc. Applying this to the cam in Fig. 4 it means that while the cam is revolving through the angle m, the tappet will have moved up to the position b, the distance ab being equal to the length of the arc xy of the inscribing circle, thus fulfilling the second condition.

The third condition means that for two-armed tappets the semi-circumference of the inscribing circle must be greater than the lift, or if r is the radius of inscribing circle and h is the lift or drop, πr must be greater than h, or approximately r must be greater than $\frac{h}{3}$, that is, the diameter of the inscribing circle must be greater than two-thirds the drop.

From the above discussion it is evident that a cam-curve designed for a certain distance between cam-shaft and stem, and for a certain drop, does not fulfill the conditions imposed when it is used for any other drop. A small range of drop is necessary to accomo-

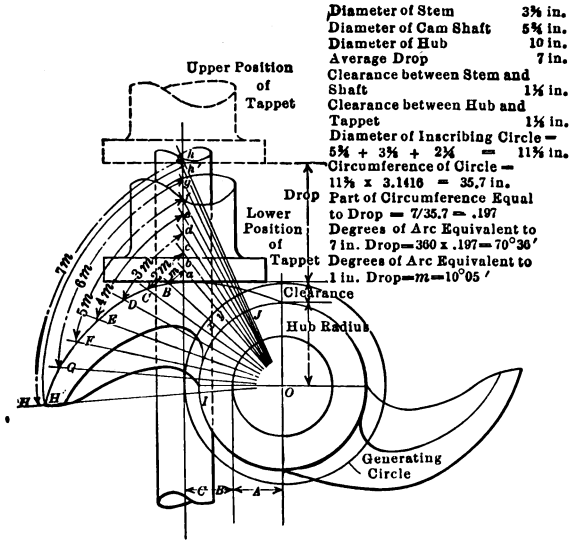
Diameter of Stem 3⅜ in.
Diameter of Cam Shaft 5¾ in.
Diameter of Hub 10 in.
Average Drop 7 in.
Clearance between Stem and Shaft 1¼ in.
Clearance between Hub and Tappet 1¼ in.
Diameter of Inscribing Circle = 5¾ + 3⅜ + 2¼ = 11⅜ in.
Circumference of Circle = 11⅜ x 3.1416 = 35.7 in.
Part of Circumference Equal to Drop = 7/35.7 = .197
Degrees of Arc Equivalent to 7 in. Drop = 360 x .197 = 70°36′
Degrees of Arc Equivalent to 1 in. Drop = m = 10°05′
Upper Position of Tappet
Drop
Lower Position of Tappet
Clearance
Hub Radius
Generating Circle
7m
6m
5m
4m
3m
2m
1m

FIG. 4. Cam Construction.

date the wear of shoes and dies, but this range should not exceed 2 or 2½ in. at the most. Of course, it is impossible to use a cam for a longer drop than it is designed for. To use it for a shorter drop means loss of efficiency since the cam must engage the tappet at a point where it is traveling with greater velocity and consequently strikes the tappet a heavier blow; the lever arm is longer than necessary and more power is required; also the cam-curve is longer for the same amount of lift near its point, than it is near the hub and the friction loss between cam and tappet is thereby increased.

A simple method of construction is shown in Fig. 4, the dimensions being about standard for a 900- to 1000-lb. stamp with a drop of 6 to 8 in. The curve upon the outer end of the cam is slightly modified to allow the upward motion to cease as soon as the tappet leaves the cam. The end of the cam is beveled to correspond to the rim of the tappet, so that the tappet is released

from an arc 2½ in. long, instead of from a sharp point, as would be the case if the end of the face were square with the sides.

The inner end of the cam-curve is filleted to meet the hub. The face of the cam is from 1½ to 3 in. wide and of about the same thickness. It is supported by a rib or web usually from 1 to 1½ in. thick. Cams are made right- and left-handed, the term being used to designate the side of the stem upon which the cam operates when viewed from the 'hang-up' position.

For attaching the cams to the shaft, keys and keyways were formerly much used. Where, as is commonly the case, a number of stamps are operated from the same shaft, this means that in order to distribute the work uniformly the cams must be ranged

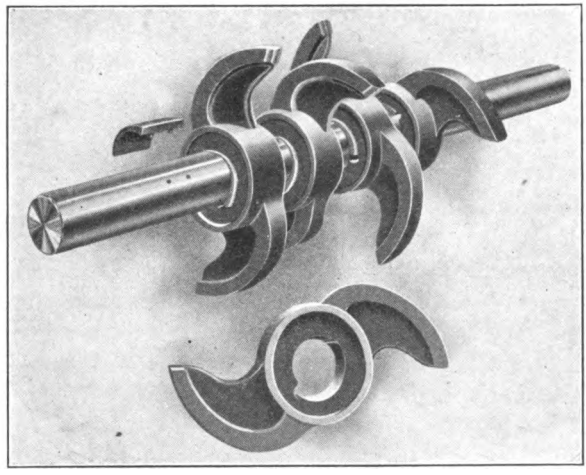

FIG. 5. Blanton Cam.

around the shaft at a uniform distance apart. For example, with five stamps to a shaft they must be set 36° apart, with ten stamps 18° apart. To accomplish this with keyed cams, each cam must have its keyway cut in a different place, and before a broken cam can be replaced, the old key must be driven out and a keyway cut in the new cam, unless a large stock is kept on hand with renewals for each position. A better way is to remove the entire camshaft and substitute another with cams in place. To obviate this difficulty of the keyways the Blanton cam was invented, and this, together with the modifications introduced by com-

peting firms, has practically displaced the old form of keyed cams in up-to-date mills.

In the Blanton cam an eccentric wedge (Fig. 5), fitting tightly in a corresponding eccentric recess cut out of the cam-hub, replaces the key and keyway. By this means the cam is secured to the shaft by a powerful friction grip, from which, however, it may be readily loosened by a few blows of the hammer in a direction opposite to the lifting reaction. The wedge is held in position by a couple of small dowel pins which fit into corresponding holes drilled in the shaft.

The Canda cam (Fig. 6) is similar in principle, having the added advantage of preventing any lateral motion along the shaft. Other fasteners of this type are on the market. One point about the circular wedges worth noting is that both the eccentric recess and the circular wedge must be made with great exactitude, otherwise the cams will not be uniformly disposed about the shaft. Therefore if cams are to be made in local shops it will usually be more satisfactory to use keys than eccentric wedges. Cams may be made of good cast-iron, in which case a wrought-iron band is usually shrunk on the hub. Special-steel cams are frequently used.

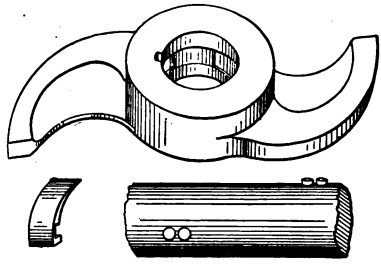

FIG. 6. Canda Cam.

Cam-Shaft. The cam-shaft (*D*, Fig. 2) is an extremely important part of the mill and should be of the best material available. It is perhaps exposed to more severe conditions than any other piece of shafting. Mild steel, chrome-nickel-steel, and hammered iron are the preferred materials. The Union Iron Works make their cam-shafts by forging down old iron propeller-shafts from large steamships. Hollow shafts have been used in South Africa and the practice is a sound one mechanically. Cam-shafts range from $4\frac{1}{2}$ to 7 in. in diameter according to the weight of the stamps. Where, as is commonly the case, the stamps are arranged five to each mortar, cam-shafts are often made sufficiently long to drive ten stamps, with the necessary overhang for the pulley.

The Drive-Pulley or Bull-Wheel (*Z*, Fig. 2) is made of wood on account of the jar and vibration to which it is subjected by the

cams striking the tappets. The wooden portions are placed radially, the thickness of the pulley being built up of several layers, secured together by bolts and a cast-iron hub with wide flanges, one of which is removable, as shown in Fig. 7. Cam-shaft pulleys vary from 6 to 8 ft. in diameter by 12 to 18 in. face. In a few instances the cam-shaft pulley is replaced by a gear-wheel. The cam-shaft boxes or bearings are mostly of the open type, and since the speed

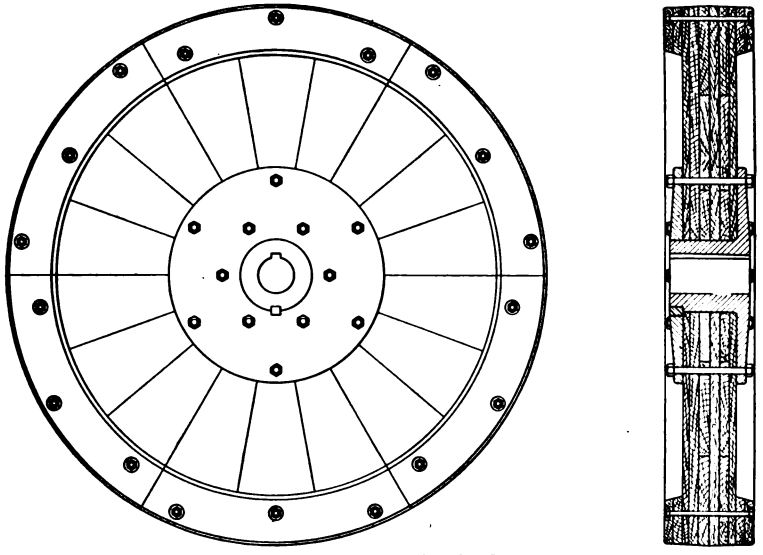

FIG. 7. Bull-Wheel.

of rotation is not great and the friction of steel on cast-iron is low, babbiting is uncommon. Covers of canvas are preferred to cast-iron as the latter frequently work loose due to the jar.

Owing to the fact that the cams operate only on one side of the stem, there is a decided lateral thrust between the tappet and the cam (Fig. 8). This thrust is lessened by running the cam close to the stem and by keeping the center of gravity of the stamp as far down as possible, *i.e.*, by concentrating the weight in the boss and shoe. This thrust is also controlled by guides on the stem, and by placing collars on the cam-shaft to prevent it moving lengthwise. A frequently used corrective is to make part of the cams left-handed and part right-handed, thus balancing the thrust from each direction.

An incidental but none-the-less important feature of cam-driven stamps is, that the stamp is partly rotated at each lift by the friction of the cam-face on the tappet. This serves admirably to equalize the wear on the shoes.

(2) THE MORTAR

The mortar or mortar-box is the vessel within which the crushing and frequently part of the amalgamation of the ore takes place. The capacity and efficiency of the stamp mill depends to a large degree upon the form and internal dimensions of the mortar. Especially is this true of the width and depth of the lower part of the mortar in the immediate vicinity of the dies.

The mortar is usually of cast-iron, although in some of the Gilpin County mills, and in cases where mills are built 'sectional' for mountain transport, the bases only are of iron, the sides and ends being of wood or light sheet-steel.

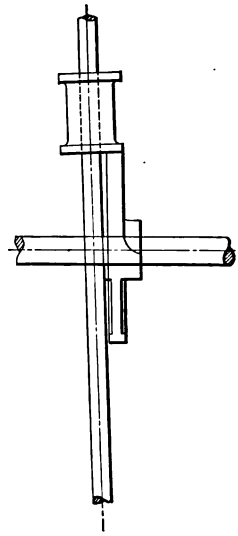

FIG. 8. Effect of Cam Thrust.

Fig. 9 shows a standard type of 5-stamp mortar, the various parts being designated by letters. *A* is the feed-throat or feed-slit, made smaller at the opening than below, in order that any piece of rock which once enters may not wedge and hang-up before reaching the dies *BB*. *C* is the screen, set in a wooden frame and held in place by the keys *D*. *E* is the chuck-block by changing which, the height from the bottom of the screen to the top of the die may be varied; this distance, designated the 'discharge' or 'issue,' is an important factor in regulating the character of crushing done by the stamps. *F* is an inside copper plate designed to catch gold within the mortar. *G*, *G*, *G*, are cast-steel liners to protect the inside of the mortar from being worn by the constant abrasion of sharp pieces of ore. *H* is the base and *I* the foundation or holding-down bolts. The base which rests upon the foundation should be machine finished to give a true bearing, as should also the faces against which the screen-frames are fitted. All other corners should be well rounded with fillets. The dies are frequently placed upon a false bottom-plate or liner. Such a mortar as shown will usually be from 4 ft. 6 in. to 5 ft. high.

The weight of the mortar is from 6 to 10 times that of each stamp; for example, in a mill erected by the writer some years ago the stamps weighed 850 lb., the mortars 5200 lb., or 6.1 times the weight of the stamp. In the mill of the Goldfield Consolidated Company, with 1050-lb. stamps, the mortars weigh 10,500 lb. The weight of the mortar must be concentrated in the base to absorb the shock of the shoe striking the die, and to give solidity to the blow struck.

If we make the discharge low, *i.e.*, if we bring the screens down close to the level of the die, other conditions remaining the same,

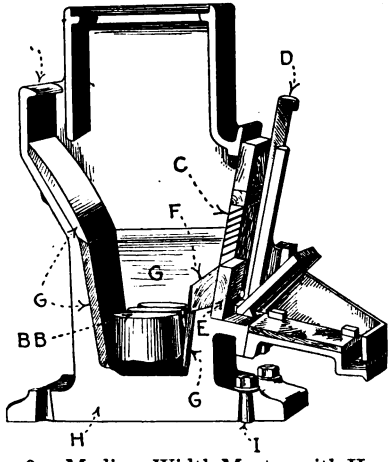

FIG. 9. Medium Width Mortar with Heavy Base for Concrete Foundation.

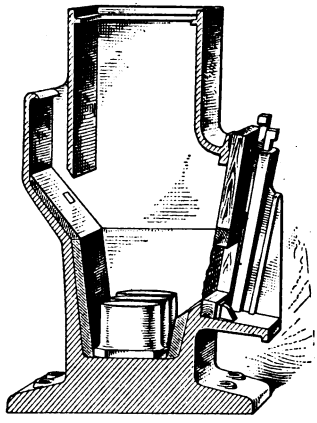

FIG. 10. Narrow Mortar for Fast Crushing.

the pulp when struck by the descending stamp will be forced against the screens more violently, and with the same screen aperture we will get a coarser product and greater capacity. Similarly as we narrow the mortar immediately above the dies, so we increase the violence of the splash of pulp when the stamp strikes the surface of the water, and again a coarser product is discharged. Thus, a narrow mortar with low discharge tends to high crushing-capacity, while a wide mortar with deep discharge gives low capacity and a finer product. These conditions should be borne in mind as they have an important bearing upon stamp-mill amalgamation to be discussed later.

Double-discharge mortars, originating in the dry-crushing silver mills, have at times been used for gold-milling. The energy of the

splash, however, seems to be lost by this arrangement and it usually results merely in an increased consumption of water, without an increased capacity of the mill. I have usually found better results to follow the closing of the back discharge in such mills.

In Fig. 9 is shown a mortar of medium width and discharge. Fig. 10 shows a narrow mortar with low discharge for fast crushing. In Fig. 11 is shown a high-discharge wide mortar, verging toward the Gilpin County type.

(3) THE FOUNDATION AND FRAMEWORK

The heavy mortar in order to serve as an anvil subject to the incessant pounding of the stamps must have an extremely solid, secure foundation. The mortar-block, as the foundation is called, has until recent years always been of timber but after much dispute it is now generally acknowledged that concrete is superior to wood for this purpose.

Mortar-Blocks may, for convenience, be divided into three classes:

(*A*) *Wooden,* (*I*) *solid,* (*II*) *built-up.*
(*B*) *Concrete.*
(*C*) *Concrete surmounted by cast-iron anvils.*

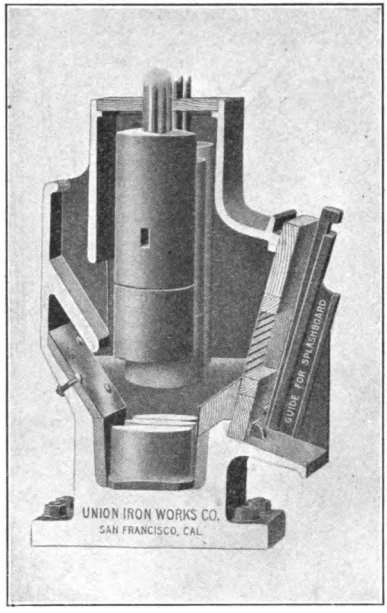

FIG. 11. High Discharge Wide Mortar with Front and Back Plates.

(*A*) *Wooden Mortar-Blocks* were for many years considered the only suitable foundation for a stamp mill, great stress being laid upon the resilience of the wooden block under the blow of the stamp. This and other claimed advantages have been proved to be largely hypothetical. Where timber is readily available, however, it is much cheaper than concrete in first cost and for a small or temporary plant is undoubtedly to be preferred.

(*I*) Solid-timber mortar-blocks (Fig. 12) are rarely of one piece of timber. For a 5-stamp battery * two timbers about 30 inches

* The unit of a mill, *i.e.*, stamps and mortar, is spoken of as a "battery," for example, "a 5-stamp battery," "a 2-stamp battery," etc.

square, or three 18 by 30 in., are bolted together, making a block from 10 to 15 ft. high when placed on end in a previously prepared pit. The bottom of this pit, which should be upon bed-rock if possible, is leveled with concrete or tailing, and the block placed on end upon this. In cases where bed-rock is not easily

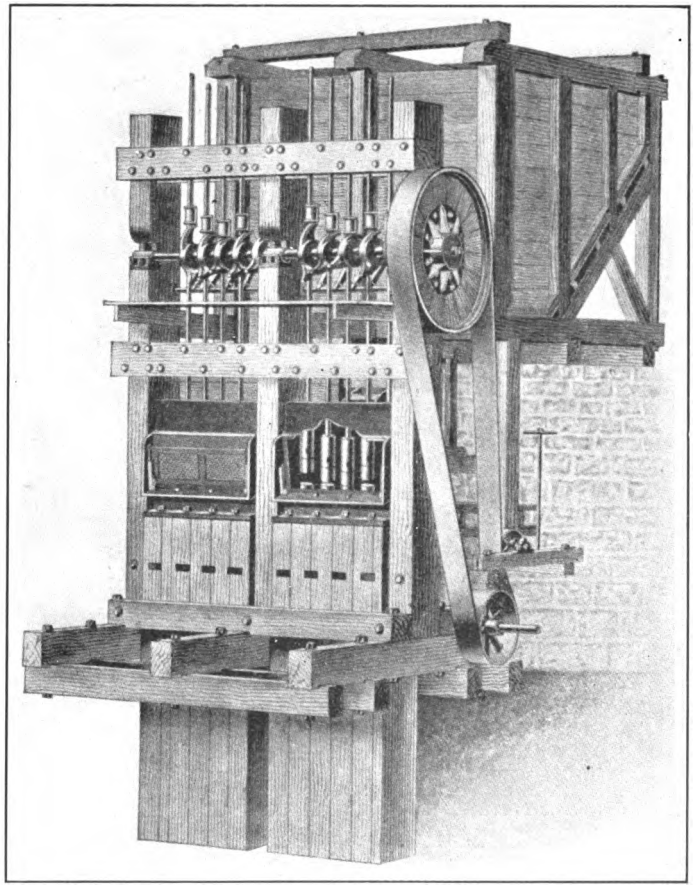

FIG. 12. Solid-timber Mortar-Blocks.

reached a foundation of concrete at least 3 ft. deep and of area proportioned to the character of the ground should be used beneath the wooden block. The sides of the pit around the block may be filled in with concrete, although tailing makes a satisfactory filling.

(*II*) Built-up wooden blocks (Fig. 13), constructed of 2 by 12 in. planks, spiked and bolted together, with joints broken, are preferred by some millmen, owing to the greater care which can be exercised in the selection of sound wood. I have used blocks of this character where large timbers were unobtainable. They are much more expensive than the solid block, and it is questionable if their use is advisable on the score of longer life as has been claimed. Where this is a consideration and cement is obtainable, concrete is a better and cheaper material.

After having been set in place, the top of the wooden block, be

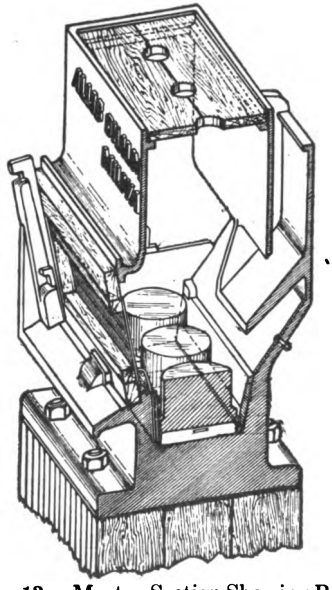

FIG. 13. Mortar-Section Showing Built-up, Wooden Mortar-Block.

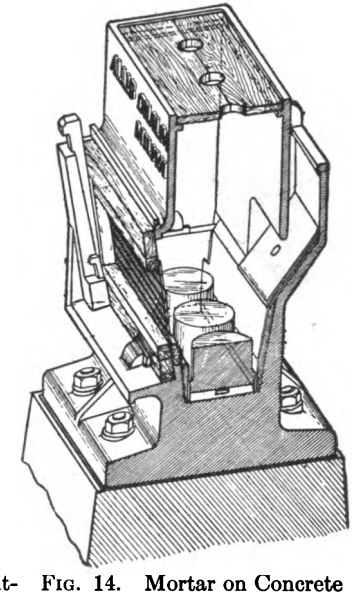

FIG. 14. Mortar on Concrete Block.

it solid or sectional, must be carefully dressed to a horizontal plane, bolt holes drilled, and pockets cut for the holding-down bolts. These bolts are usually from $1\frac{1}{4}$ to $1\frac{3}{4}$ in. in diameter and about 2 ft. 6 in. to 3 ft. long. Between the mortar-base and the top of the block it is usual to place several layers of tarred blanket or strips of rubber belting. Some manufacturers supply a felt pad and others a mat of $\frac{1}{4}$-in. rubber for this purpose.

(*B*) *Concrete Mortar-Blocks* seem to be rapidly gaining in favor and it is now exceptional to find a large mill in process of

erection provided with wooden blocks. It was for a long time feared that increased stem breakage and other difficulties would follow the use of the concrete mortar-block, but this does not seem to have been proven. Crumbling of the concrete beneath the mortar base has been remedied in two ways, first, by making the mortar base wider, and second, by interposing some sort of cushion between the mortar and concrete, both to give an even bearing and to absorb part of the impact (Fig. 14). Most mills use a thin

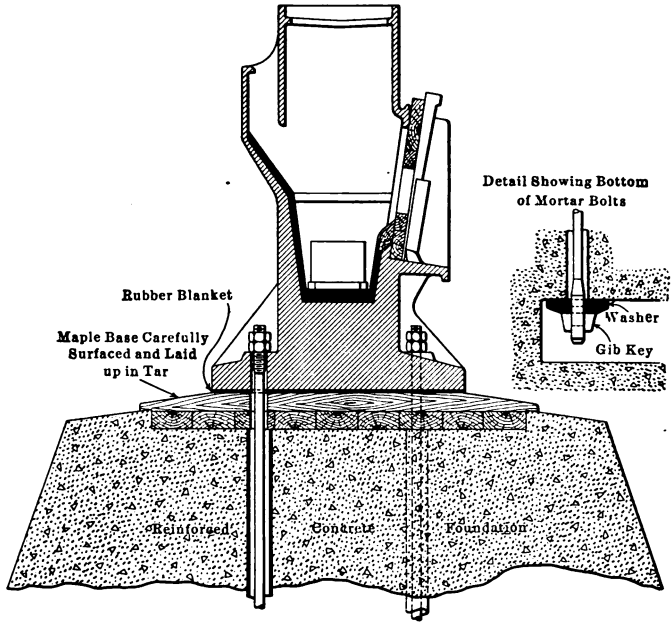

FIG. 15. Mortar on Concrete Block with Wooden Cushion.

rubber pad or mat; others prefer a cushion of hard wood upon the concrete (Fig. 15). Another difficulty has been to make the holding-down bolts secure; this has been accomplished by placing them in a pipe bedded in the concrete, leaving both ends accessible. The lower ends of the bolts are made to bear against a 4 by 6-in. timber instead of against the concrete. As in the case of the wooden blocks, if bed-rock can be reached so much the better, but any reasonably solid ground is sufficient, provided the area of the base of the foundation is made great enough. The advantages of permanency, solidity, and consequent greater stamp-capacity are

all with the concrete mortar-blocks and their use promises to become practically universal.

(*C*) *Concrete Mortar-Blocks with Cast-iron Anvils* have many advocates. Two-thirds of the batteries in the great stamp mills of the Alaska Treadwell Company are mounted on this type of foundation, supplied by the Allis-Chalmers Company, see Fig. 16. (Also shown in elevation in Fig. 1, page 1.) The Gold King Company's mill, probably the most up-to-date plant of its kind in Colorado, has iron anvil-blocks, built on the principle described by the Denver Engineering Works Company, as a 'sub-base' block (Fig. 17 and 18). The anvil-block offers a means of having a base of ample width and weight for the mortar without the initial stresses and strains which would be established in casting a base of similar proportions as a part of the mortar. The same means of securing the sub-base or anvil-block to the concrete should be employed as would be used in the case of a mortar setting directly on the concrete.

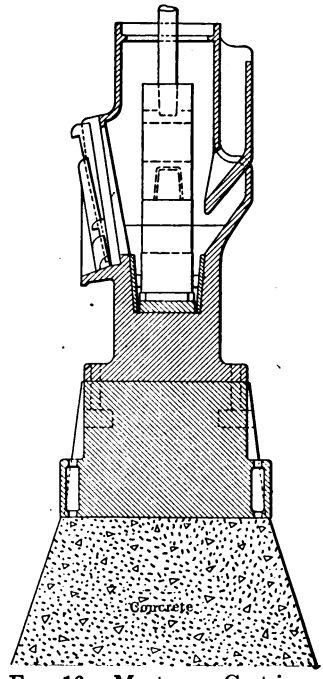

FIG. 16. Mortar on Cast-iron Anvil Block.

The Frame is designed to support the cam-shaft and to guide the stamp stems as they rise and fall; frequently also, but not necessarily, the main line-shaft is carried thereon. The great majority of battery frames are built of timber, although frames of structural steel (see Fig. 18) are gaining in favor and cast-iron frames for small installations are often seen.

All wooden frames have in common the large vertical post on each side of the mortar, called the battery-post, usually 12 or 14 by 24 in. in section and from 18 to 20 ft. long. Where wooden mortar-blocks are employed, these posts rest on cross-sills each side of the mortar-block; in the case of concrete blocks a cast-iron base, frequently provided with a rubber cushion, takes the foot of the post. My attention has been recently called to the fact that

this is frequently a point of failure in stamp-battery frames. The base or shoe should be heavy, and the post should be secured to it by a stout bolt on each side, attached to the post by a solid lug or bracket. Between the battery-posts run the girts, to which the guides are secured. The cam-shaft boxes or bearings are usually set in niches cut in the battery-posts, about two-thirds of their length from the bottom. In accordance with the manner in which

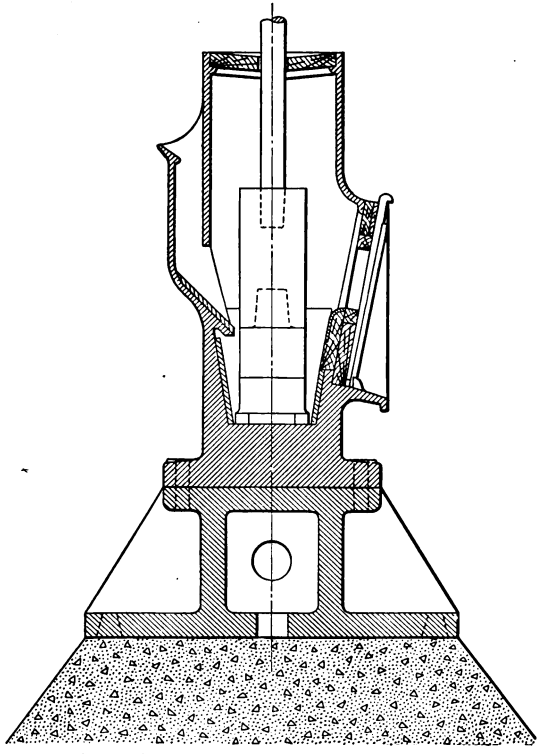

FIG. 17. Mortar with Sub-base Block.

these battery-posts are secured to, and supported by, the other members, the types of frames are classified.

The *A*-frame (Fig. 19) is used for small portable installations. It is not convenient where plates are used, but is light and inexpensive.

Frames of either the back-knee or front-knee type are the most commonly used. The back-knee (Fig. 1) is the one preferred in

the larger plants. It gives greater light about the plates and a full view of the mill from the front of the batteries. The line-shaft, however, is not so well placed as in the case of the front-knee frame. Many mills are practically a composite between the two types, having a front-knee frame connected with bins behind the stamps (Fig. 20), or more rarely a back-knee frame connected with the posts of the building in front. By placing the line-shaft level

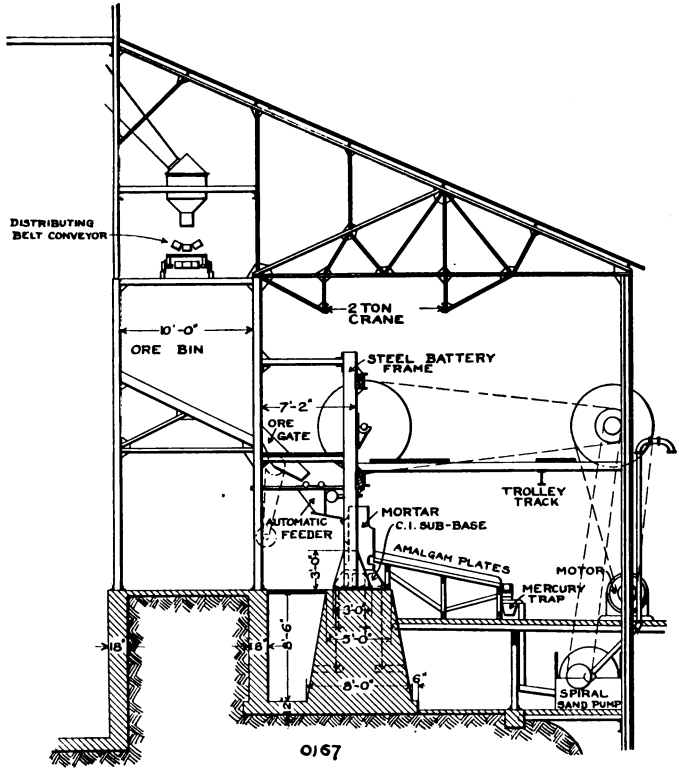

FIG. 18. Steel Battery-Frame.
(Courtesy of Denver Engineering Works.)

with the cams, a horizontal belt-drive is secured, but the belts are in the way and light is sacrificed. Most large stamp mills are driven from a line-shaft behind and below the mortar, a belt-tightener being used to prevent slipping. This also serves practically as a friction-clutch, since upon releasing the strain of the tightener the belt slips on the drive-pulley.

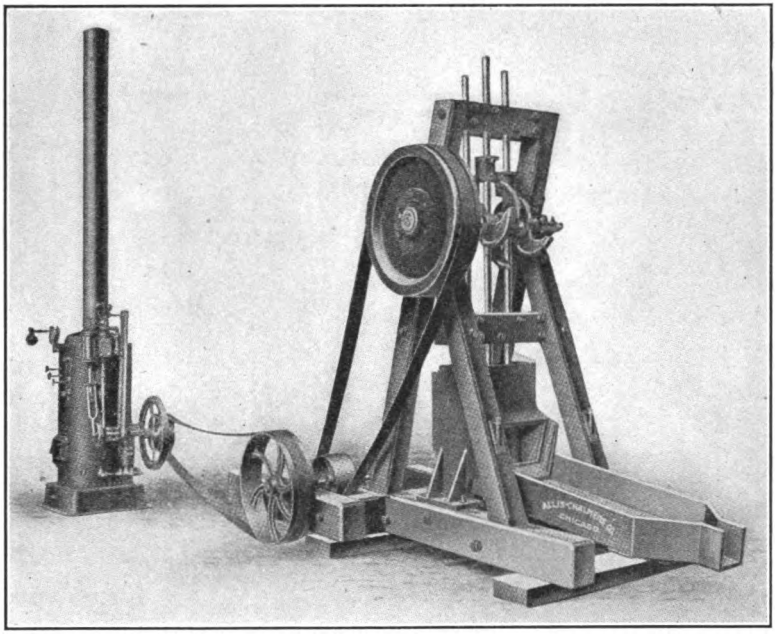

FIG. 19. "A"-type Battery-Frame.

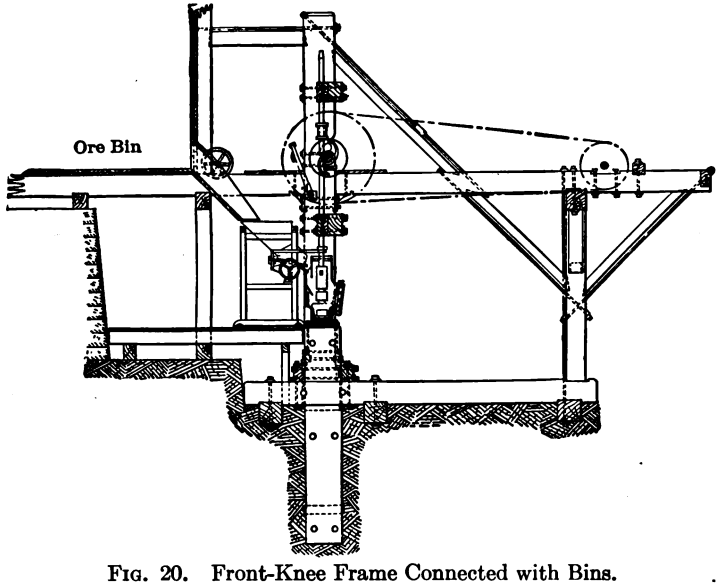

FIG. 20. Front-Knee Frame Connected with Bins.

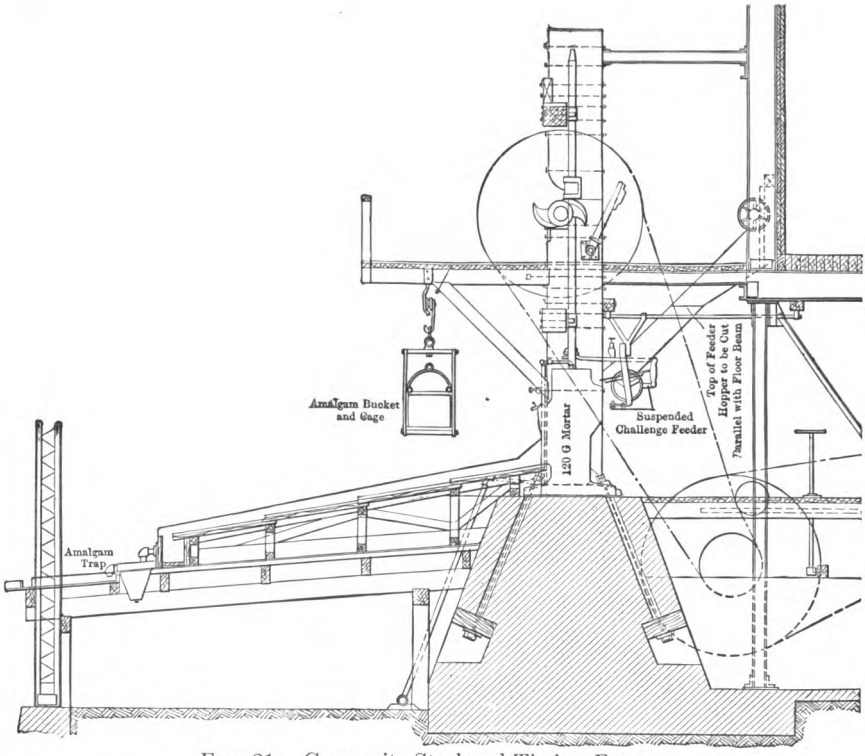
Amalgam Bucket and Cage
120 ℄ Mortar
Suspended Challenge Feeder
Top of Feeder Hopper to be Cut Parallel with Floor Beam
Amalgam Trap

FIG. 21. Composite Steel and Timber Frame.

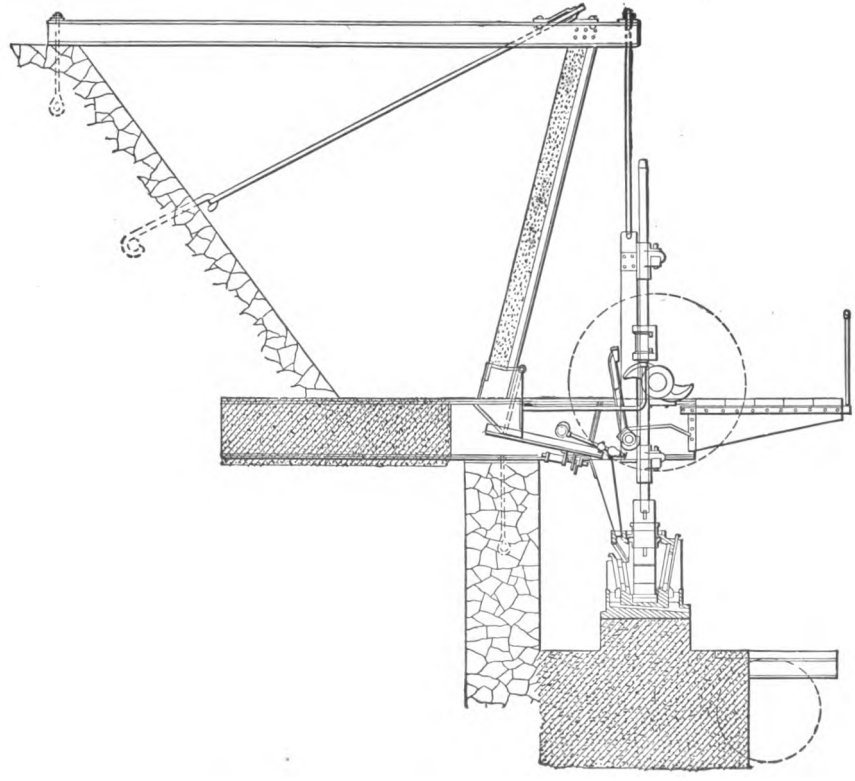

FIG. 22. Boss Cantilever Battery-Frame.

These various types of frames are, of course, subject to modification and may be combined with each other or with the ore-bin framing in various ways. Where not otherwise provided for, brackets are secured to the battery-posts in front to support the platform or runway giving access to the cams, cam-shaft, and boxes.

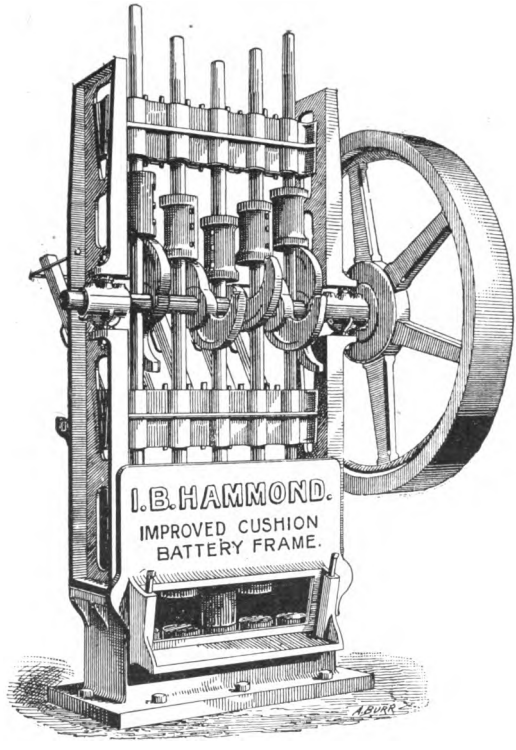

FIG. 23. Cast-iron Battery-Frame.

Steel frames are not at the present time in extensive use. The incessant jar and vibration to which the battery-post is subjected is considered sufficient reason for the non-employment of a metal so elastic as steel for this member. This objection could undoubtedly be overcome by the employment of a sufficiently heavy section, but, except for durability, it would offer little advantage over the wooden post, particularly in the matter of weight. There is no reason, however, why steel should not be used for the balance of the frame, including ore-bins, and the Goldfield Consolidated

frame, shown in Fig. 21, is as satisfactory a scheme as can be devised at present. A steel frame on the cantilever principle (Fig. 22), in which I-beams projecting from the bin foundations form the supports for the cam-shaft, thus eliminating battery-posts and all

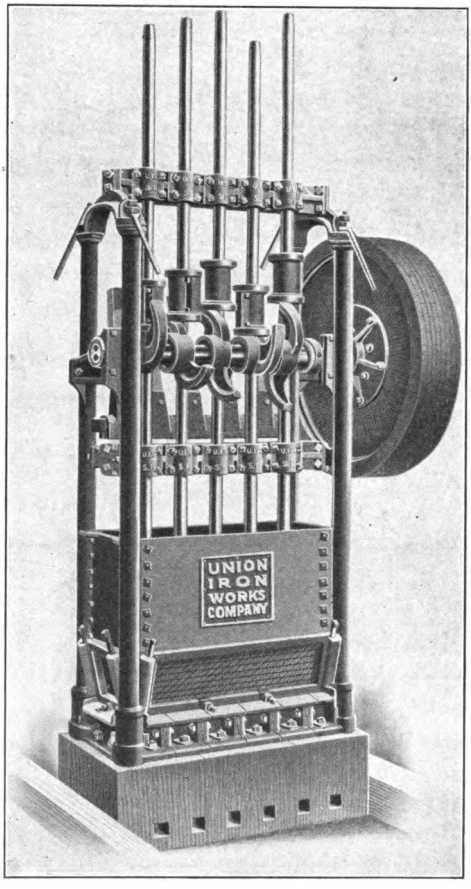

FIG. 24. All-iron Frame Battery in Sections.

other obstructions about the battery, has not proved entirely satisfactory.

Cast-iron frames are used in the United States for small portable mills and they are admirably adapted for this purpose (Fig. 23). A frame with cast-iron columns and brackets, tied by steel tension bolts, was furnished for a 100-stamp mill in Bolivia by the Union

Iron Works Company (Fig. 24). Wherever timber is scarce this form of frame would seem to be commendable. Cast-iron frames and heavy columns without tie-rods are extensively used in Australia. In the City Deep Mill in South Africa reinforced-concrete pillars have been substituted for the usual battery-posts. On top of the pillars rests a heavy girder which carries the guides on its

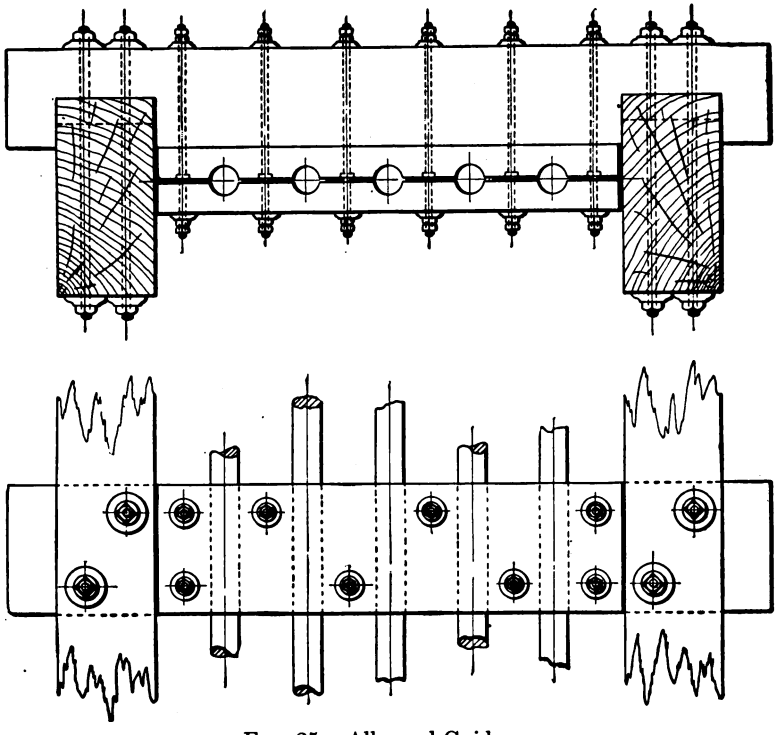

FIG. 25. All-wood Guides.

lower edge, and on the upper side are set cam-shaft bearings between each cam and its neighbor.

Guides. The function of the guides is to prevent the stems from deviating from their vertical path when being raised by the cams. They are, of course, a hindrance to the stamp in falling and unless they are kept reasonably loose and well lubricated, interfere seriously with the efficiency of the blow struck by the stamp. Guides are made either of wood or iron, or of a combination of both of these materials. Two sets of guides are used; the

first is placed about 6 in. above the highest point reached by the tappet, the second, or lower guide, is set as close to the top of the mortar as will allow access thereto, the lower the better. A

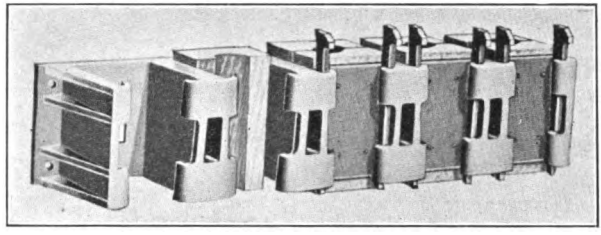

FIG. 26. Sectional Wooden Guide in Iron Frame.

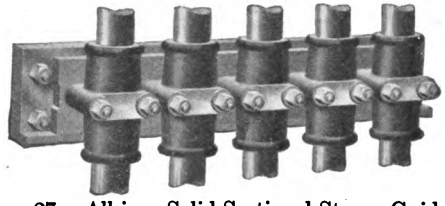

FIG. 27. All-iron Solid Sectional Stamp Guides.

satisfactory guide should allow adjustment in all directions. This is necessary to take up the wear due to a thrust, which is the re-

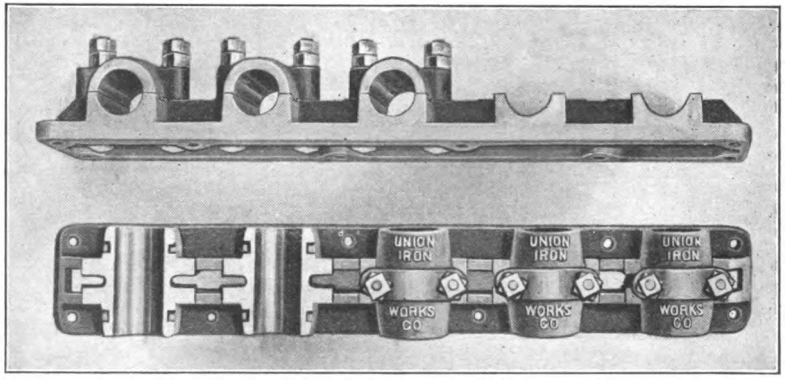

FIG. 28. All-iron Guides.

sultant of the direct thrust of the cam-face and of the eccentric lift which the cam gives to the tappet.

The simplest form of wooden guide consists of two hardwood

timbers bolted to the guide girt, with semi-cylindrical grooves cut in the face of each timber (Fig. 25). This guide is much used, its simplicity compensating for the lack of endwise adjustment. Adjustment laterally is provided for by starting with shims, or liners between the guides; these are gradually removed as the wood wears away, and later the faces are planed off as the stems continue to cut into the wood.

Individual wooden guides are used to some extent, see Fig. 26. These permit adjustment of separate stems without stopping the

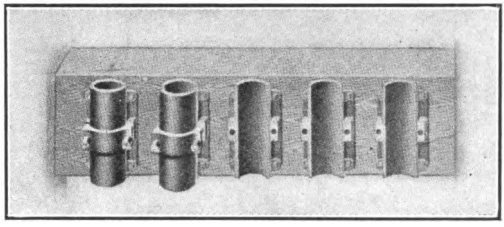

FIG. 29. Iron Guides on Wood Girt.

entire battery. All-iron guides are employed in some of the larger mills, both plain and babbitted boxes being common; see Fig. 27, 28 and 29.

(4) ACCESSORY APPARATUS AND OPERATION

" Hanging-up " stamps. For the purpose of supporting the stamps so that the tappets shall be clear of the cams, finger-bars or latch-fingers, as they are called, are provided (*G*, Fig. 2); these are shod at each end with iron and provided with hand grips in front. They are pivoted on the jack-shaft (usually of about 3 in. diameter), which in turn is carried on brackets on the inner side of the battery posts. When the stamps are to be hung up, a cam-stick (Fig. 30) is placed on the cam, thus causing the stamp to rise higher than usual, and permitting the latch-finger to be swung under the tappet, which is thus kept at a height beyond the reach of the cam. Various ingenious mechanical devices have been used at times to replace the old-fashioned cam-stick and latch-finger, but none of these has found general acceptance.

The Overhead Track and Crawl, to which is attached a chain-block, is indispensable for picking up stamps requiring repairs, for handling cam-shafts, and other heavy lifting on the cam floor.

The Screens, in conjunction with the height of discharge, regulate the maximum size of the crushed product turned out by the stamps. Upon the mesh * and character of the screen used depends, to a large extent, the capacity and the extraction of the mill. Screens are of two classes, punched and woven.

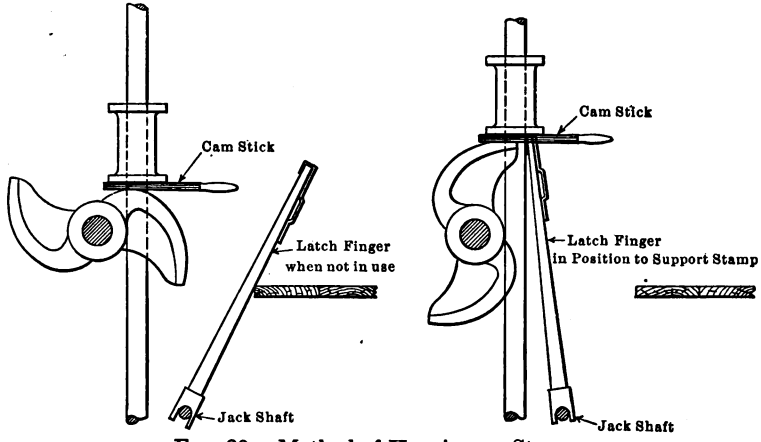

FIG. 30. Method of Hanging up Stamps.

Punched screens, as their name would indicate, are made by punching holes in thin metal plates. Steel or iron is usually the metal, but copper 'gratings,' as they are called, are used in Australia. Some American millmen attach great virtue to tin-plate screens from which the tin is burnt in the forge. Punched screens are designated by the shape of the hole punched in them. 'Round-hole' and 'slot' designate the shape of the hole; 'burr-punched' denotes the plate is indented in the cutting, 'clear-punched' that the piece is taken clear out, leaving no indentation. Burr-punched diagonal-slot screens are great favorites in some camps, the burr being placed inside, with the idea that as the screen wears the indentation will flatten out and thus keep the size of the opening from growing larger. Horizontal and vertical slot screens, both clear and burr, may be purchased. Punched screens are designated as to the width of their apertures by an arbitrary system of numbers, for common sizes if x = trade number, $(x - 1)\ 5$ = linear mesh.

* In American practice screen aperture is designated by the number of holes to the linear inch, but in South Africa by the number of holes to the square inch. Ten mesh in Goldfield is 100 mesh in Johannesburg.

Woven screens are made of wire; brass or iron being the usual material. In all but the finer meshes the diameter of the wire and the diameter of the hole are practically equal, thus making the ratio of opening to total screen average about 25 per cent.* Brass-wire is preferable as it does not rust, but its use is out of the question where quicksilver is fed into the battery.

Comparing punched and woven screen the following points may be noted:

Punched screens wear longer, do not bag or become distorted in shape, have a smaller percentage of opening, and deliver a less uniform product than wire screens. In general, high capacity demands the use of cloth or woven screens with consequent higher screen-cost per ton than would be the case if punched screens were used.

Screen Frames are practically always of wood. In a five-stamp battery the screen opening is from 10 to 20 in. wide by 48 to 50 in. long. With wire-cloth screens of fine mesh (20 or under), the opening is divided into two, or sometimes three, panels by vertical bars, to support and stiffen the cloth. With punched plate screens this is unnecessary. Screens are tacked to the inner side of the frame, which is made secure and tight against the planed faces of the mortar. To accomplish this the inner edges of the frames are covered with belting or blanket, and the frames are held in place by keys at each side of the mortar, see *D* in Fig. 9. Keys at the bottom of the frame, on the mortar lip, are also used. Screens are sometimes placed in separate panels, thus making a double frame.

Provision should always be made for a handhole above the screen or higher up on the sloping front of the mortar, for the removal of chips and other 'trash' which will accumulate against the screen, impairing its efficiency and frequently causing breakage. Screens usually incline outward from 10° to 15° from the vertical.

Splash-Boards or Canvas Curtains are frequently placed in front of the screen to confine the spray from the mortar. Where a board is used it should incline outward at about 45° to allow ready inspection, and should come to within about an inch of the screen frame along the bottom edge. Such a board covered with a copper plate forms an efficient gold and amalgam catcher.

* See page 113 for table of apertures and wire diameters in standard screens.

Housing or Cover Boards are placed at the top of the mortar, resting on ledges cast for the purpose. They are made in two pieces with notches cut along the inner edge to fit the stems, and serve to confine the splash within the mortar.

Rock-Breakers have gradually come to be regarded as indispensable prerequisites for the treatment of ore by stamp milling.

It is not proposed to enter into any description of the various types of breakers, but merely to point out that the stamp mill is not designed to crush coarse rock and that to attempt to make it do so is fatal to its capacity and efficiency. Nevertheless, it seems to be possible to go too far in the opposite direction, and some millmen have found that beyond a certain point there is no gain but an actual loss in too fine crushing previous to stamping.* It is undoubtedly true that there should exist a definite ratio between the weight and drop of the stamp and the diameter of the particle to be crushed. Present knowledge would seem to indicate $\frac{1}{10}$-in. diameter for each 100 pounds of stamp-weight, as about right for average ore, standard drops, and fine crushing.

Breakers of the Blake and gyratory types have displaced all others. Gyratories are gaining favor as they require less attention in feeding; and for the same reason the grizzlies or sloping bar-screens, formerly so much in evidence, are going out of fashion, and are being replaced where necessary by shaking or revolving plate-screens or trommels.

Rolls in some few instances have been included as part of the accessory crushing plant for stamps, but the practice is unusual. As a preliminary to certain other crushing devices, to be mentioned later, they have attained a certain degree of popularity. Their preëminent field is, of course, that of fine and intermediate crushers for concentrating mills.

The literature on rolls is voluminous and should be consulted if further information is desired.

Ore-Feeders. In the early days of stamp milling all feeding was done by hand, and even after the introduction of mechanical feeders, it was still claimed that hand-feeding was the more efficient. In Gilpin County, where, strange to say, hand feeding is commonly used, I have seen the feeder shovel in a hundred pounds or so of rock and go home, to his dinner, leaving the battery to "pound herself out" as he expressed it. In all mills making any pretense

* W. Beaver, J. C. & M. M. Soc., S. A., Vol. 6, page 215, Jan., 1906.

to efficiency, however, automatic ore-feeders are in use. These feeders, as long as no marked variation occurs in the character of the ore, will work day and night, practically without attention, giving maximum capacity, requiring almost no repairs and saving the wages of an attendant for every 20 stamps. Feeders are of two general types; both of these, at one time, were patented, but they have long been open to manufacture by any one.

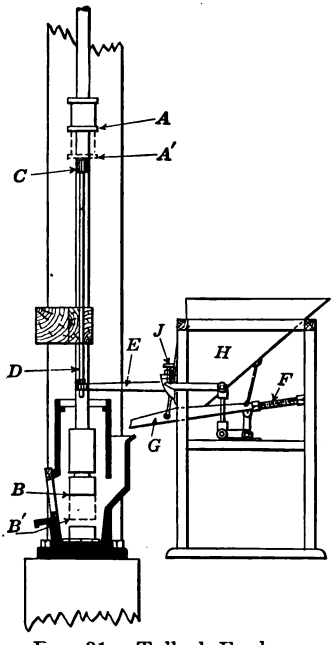

FIG. 31. Tulloch Feeder.

As soon as the tappet *A* reaches *A'*, the shoe *B* will have reached *B'*; the lower edge of the tappet then engages the bumper *C*, pushing down the rod *D*; this in turn depresses the end of the rod *E*; by this means the system of levers jerks the tray *G* backward and the contents of the tray are spilled into the mortar. The spring *F* returns the tray to its position. The hopper *H* carries the supply of ore and feeds to the tray *G*. *J* is the hand wheel which controls the position of the rod *E* and hence the height of *C*.

Tulloch Feeder. The simplest feeder is known as the Tulloch, although for the principle upon which it operates, we are indebted to C. P. Stanford, a Californian, who used it in 1855. It consists of an inclined tray, set beneath a hopper. A reciprocating motion is given to the tray by means of a system of levers connecting with a bumper-rod actuated by one of the stamps, usually the center one. By reference to Fig. 31 it will be seen that as ore accumulates on the die, the tappet will not descend far enough to strike the bumper and the feeder does not work; as the supply on the die is crushed and discharged through the screen the shoe descends to a lower point, and consequently the tappet striking the bumper causes the ore to be delivered by the feeder. Although extremely simple in operation, the Tulloch or Stanford feeder will only work with the driest and most granular ore; wet, sticky, or talcose material often hangs in the tray, and is not dislodged by the blow of the stamp on the bumper.

The Challenge Feeder (Fig. 32) has, for the reasons cited, practically replaced all others. Originally built and patented by Joshua Hendy of San Francisco, this feeder, modified in recent years as to the details of the operating mechanism, is without a serious rival. Like the preceding type, it is operated by a blow from the stamp, the amount of the blow being regulated by the quantity of ore within the mortar. In this case, however, the hopper feeds to an inclined circular cast-iron plate (*a*), the revolution of which carries the ore to the mortar. The underside of this revolving plate gears with a bevel pinion (*e*) which is revolved by a system of levers (*m,n,p,k*), connecting the bumper (*r*) with a friction ratchet (*h*) mounted on the same shaft as the pinion. As the ore is

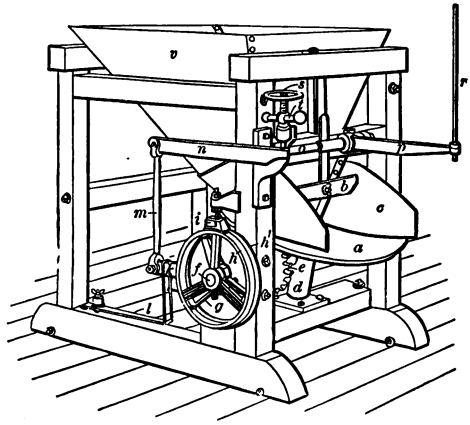

FIG. 32. Challenge Feeder.

brought around by the plate it is scraped into the mortar by the wing plates (*c*). The amount of ore to be fed is regulated by the hand wheel (*s*) which raises or lowers the bumper, thus causing the tappet or collar to engage it sooner or later as the case may be. In place of the rod and rubber bumper of the Tulloch feeder operated by the tappet, the newer forms of Challenge feeders are driven by a collar on the center stem striking against a forked bumper-rod. The suspended feeder (Fig. 33) is hung from a light track on the underside of the cam floor, thus leaving a clear space behind the battery.

The great superiority of the mechanical feeder lies in its perfect regularity of action, and in the automatic adjustment of feed to depth of layer of ore on the die. The thinner this layer,

other things being equal, the less cushion there will be to the blow struck by the shoe and the more effective will the blow be. Evidently this depth cannot be less than the diameter of the particles to be crushed, and this, as already pointed out, should depend upon the weight of the stamp. The importance of this is recognized by

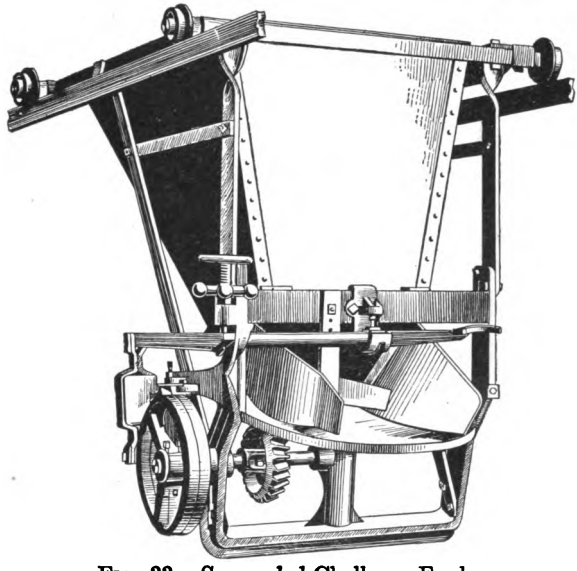

FIG. 33. Suspended Challenge Feeder.

all good millmen. I have had men on opposite shifts, when the competition was keen, change the adjustment of the feeder when going off shift, to make the layer of ore deeper and consequently to cut down the tonnage put through by the other shift. A good millman will, of course, detect this by taking hold of the stem and thereby judging of the nature of the blow being struck. The blow if dead indicates too heavy feed. If "iron strikes iron" the feed is too light. The right blow is soon learned and an educated ear can detect from some distance whether conditions are right in any particular battery.

Water Service. An ample water-supply is one of the most important items in stamp milling work. Where water is scarce, as is often the case in the arid regions of the southwestern states, the mill tailings are discharged into impounding systems or settling tanks and the supernatant liquid — for in extreme cases it can scarcely be termed water — is used over again in the batteries.

In any event, of course, an adequate piping system is necessary, and the water-supply must be under a constant head to avoid the necessity of continually adjusting battery-cocks, etc. The main for the stamp-service is usually laid in front of the batteries just below the cam-floor level, and pipes for each battery lead from it. Occasionally this main service pipe is placed below the plate-floor, in front of the mortar blocks.

As to the manner of feeding the water into the mortar itself, a variety of practices is seen. There are three places where water may be introduced.

(1) *Through holes in the housing boards.*

(2) *Through the feed throat.*

(3) *Through special openings usually at, or below, the level of the dies.*

When the first method is used, some millmen prefer one pipe as near the center stamp as possible; others place a pipe beside every stem. The majority following this method, however, use one pipe between stems No. 2 and 3, and one between No. 3 and 4. In some mills these individual jets are pointed straight toward the back of the mortar.

A single pipe or even a piece of hose pushed a short distance into the feed throat has many advocates, and occasionally a pipe at each end of the feed throat is seen.

In a Nova Scotia mill, Richards describes the use of a jet in front of and between each pair of dies, and in a recent South African mill individual jets are placed behind each die.* In every case it is essential that there be a suitable hose connection for every battery or pair of batteries for washing down plates, hosing-off lip of mortar, and for general utility.

The flow through the smaller individual pipes about the battery is best controlled by double cocks, one near the battery to regulate the flow, the other to turn water off and on; in this way adjustment is not necessary every time the mill is started. Globe and gate valves will not stay set, owing to the vibration.

The water fed to the battery has two functions: (1) To give a medium that will aid the splash, and hence the discharge of the battery. (2) To dilute and give mobility to the pulp, and to allow separation of gold from gangue, both inside and outside of the mortar. The quantity used varies with the character of the ore, the fineness of crushing, and the slope of the outside plates. The

* E. M. Weston, Eng. & Min. Jour., Vol. 89, page 172.

amount is stated in various ways, tons of water per ton of ore being a common method of measurement. Richards states that twenty-five mills throughout the United States show an average of 6.17 tons of water per ton of ore. Gallons of water per stamp per minute is a popular but decidedly unsatisfactory method of expressing the water consumption. The quantity varies from as low as 1.3 gallons in some of the Gilpin County mills to as high as 6.6 gallons in the South African mills. Other things being equal, the water consumption of course depends upon the stamp duty and varies with it. This consumption, as has been already stated, may be decreased, as far as supply of new water is concerned, by settling-tanks and the like. In one case I was able by this means to keep a stamp mill operating for some time on about 0.5 gallon of fresh water per stamp per minute, corresponding to 2.1 tons of new water per ton of ore.

For transposing from one system to the other it is convenient to remember, that gallons per minute multiplied by six equals tons per day.

Starting the Mill. Presuming the battery-frame and guides for each battery to be in place, the stems, with the tappets keyed lightly in their approximate positions, are lifted into their places with the aid of the crawl and chain-block, and the caps are placed loosely on the guides. The latch fingers are set under the tappets. If not done previously, the dies should now be placed in the mortar with $\frac{1}{2}$ to $\frac{1}{4}$-inch of clean sand beneath them to make an even bed. If false dies or bottom liners are used sand should be placed below and above them also. The dies should then be packed about with sand and fine rock to keep them in place until the ore has bedded itself about them, although this precaution is often omitted.* A 2-in. plank is then put on top of the dies and one boss at a time is set on the plank and in line with its stem, which is lowered into the socket in the boss, and tapped at the top end with a heavy copper hammer to drive it home. (An ordinary sledge hammer may be used if care is taken to protect the stem from abrasion.) This is repeated for each stamp until all stems are set. To make more secure, the shoeless stamp is raised and dropped several times, allowing the boss to strike *on the plank*. The shoes are set in place, and wedges, as described, ranged round the shanks; each boss is dropped over its corresponding shoe shank and thus picks up the shoe which is driven home by several drops on the plank as before.

* Setting the dies in place with concrete is an interesting departure that seems to promise well.

A few shovels of ore are placed on the dies, and the battery is run slowly or by hand for a moment or two till all is secure. The height of drop is then arranged by placing a block of wood, the height of the proposed drop, on the die and lowering the shoe upon it.* The corresponding cam is then revolved until its point is at the top center, and the tappet is set resting upon it and the keys driven home.

If inside plates are used they are sprinkled with mercury and set in place, and screen frames are put in and keyed. The feeders are inspected, the water-feed is looked over, and as soon as the outside plates are in condition and the rest of the mill is in shape, the stamps are ready to begin crushing ore. When all these points have been attended to, the millman starts up one battery at a time, dropping first the center stamp as this works the feeder, and then the two others on each side of it and lastly turning on the water-feed. The water-feed is then adjusted until the proper wave motion is seen on the plate; the feeder is adjusted by the 'feel' of the stems until the minimum feed which will prevent the shoes from striking the naked die is found. If the ore is inclined to bank under one stamp, that stamp is given a slightly greater drop by lowering the tappet; or if one stamp constantly 'pounds iron,' its drop is decreased by setting the tappet higher on the stem. As shoes and dies wear away the drop becomes greater and thus compensates for the loss in weight. It is usual to correct the greater drop by adjustment of the tappets.

Lubrication. The stamp mill in common with all other forms of machinery requires lubrication, but the fact that grease is the arch enemy of both amalgamation and concentration, renders this a peculiar problem. There are two solutions. Either make it impossible, by extra precautions, for oil or grease to get into the battery, or else substitute something in place of grease or oil as a lubricant. The first method is the most commonly used; drip boxes are placed under all shafting, oil wells are provided on cam-shaft bearings and a canvas curtain is hung between the cam-shaft and the top of the battery.

Among the substitutes for grease, compounds of soft soap and graphite, or of molasses and graphite, are urged by some authori-

* In regular operation the height of drop is easily regulated without shutting down other stamps or opening the mortar, a chalk-mark on the stem, or even the ring of lubricant which marks the lowest point of the stem-travel in the guide, giving all the index required.

ties. In warm weather the latter compound is excellent for attracting flies and other insects, so that in a short time the cams and cam-shaft bearings look like a heavily-loaded, sticky fly-paper. As in all other work the least amount of lubricant which will suffice should be used.

The Order or Sequence of Drop is an important factor in the capacity of a five-stamp battery. Both the splash on the screen and the distribution of the ore on the dies are largely affected by it. It is usually conceded that adjacent stamps should not fall in succession and that as one stamp is falling its neighbor should be striking. The wave motion produced should not result in carrying the ore to one end of the battery.

The California drop is much in favor. Numbering the stamps from left to right, the sequence is 1–4–2–5–3. This fulfills the condition imposed and does not give rise to any marked accumulation of ore at either end. A sequence known as the Black Hills drop, 1–3–5–2–4, has many advocates and discussion as to the comparative merits of each order has at times waxed warm. It has been pointed out by J. E. Collie * that one of these is simply the left-handed version of the other.

California numbers 1–2–3–4–5.
Designated by letter A–B–C–D–E.
Black Hills numbers 5–4–3–2–1.

Using letters in place of numbers and carrying out the sequence we get:

California drop A D B E C A D B E C
Black Hills drop E C A D B E C A D B

so that it is not easy to see why the fact that one man counts from the left, and the other from the right should have any marked effect upon the efficiency of the mill.

The Gilpin County drop, 1–5–2–4–3, violates the law of non-adjacent sequence. It is quite common, however, especially in the 'slow-drop' mills of the district from which it takes its name. The perfect symmetry of the arrangement is attractive. T. A. Rickard says that this sequence promotes rapid crushing, but causes scouring of the inside plates.†

* Pacific Miner, Vol. 16, page 50.

† Stamp Milling of Gold Ores, 1st Ed., page 89. (This may be true in high-speed mills with heavy stamps; it of course neither promotes rapid crushing nor causes scouring in Gilpin Co.)

Other sequences are used elsewhere but are rarely seen in the United States. It is said that in many of the Australian mills the stamps drop in pairs simultaneously, 1 and 5, 2 and 4, followed by the middle stamp.

Height of Drop. The height of drop multiplied by the weight of the stamp is roughly constant, that is to say a light stamp is used with a long drop and a heavy stamp with a short one. With the average stamp weight of 1000 pounds a drop of 5 to 7 inches is common.

' Speed ' of Drop. The number of drops per minute depends, of course, upon the weight of the stamp and distance through which it falls. The acceleration of gravity, retarded by the friction of the guides, determines the minimum interval between blows. Any attempt to lessen this interval results in 'camming,' that is, in the

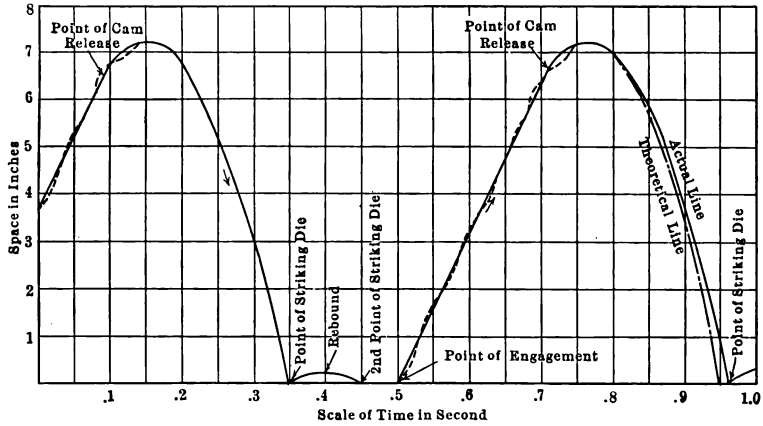

FIG. 34. Morison Indicator Diagram for 900-lb. Stamp dropping 6½ in. 97 times per Min.

cam catching the tappet before its descent is complete. D. B. Morison, an eminent English mechanical engineer, made an exhaustive investigation concerning the velocity of the stamp in its rise and fall and has computed from the data thus obtained, a table showing the maximum number of drops possible with any given height of drop.*

The inscribing circles given in the table with the shorter drops are less than the sum of the diameters of stem and cam-shaft plus the clearance. No account is taken of the fact that the friction of the guides is a variable quantity, dependent upon conditions of

* Richards, Ore Dressing, Vol. 1, page 216.

lubrication, and that the guide friction will retard a heavy stamp less than a light one. With well lubricated guides the writer has run a battery of 900-lb. stamps with 6-in. drop at the rate of 115 drops per minute with only an occasional 'camming.' In the chapter on "Variations in Practice" will be found a number of examples of height and 'speed' of drop. Fig. 34 shows an indicator diagram taken by Mr. Morison which represents graphically the cycle of progress in the rise and fall of the stamp.

Power. The power required for driving stamps is used in two ways: (1) In raising the stamps, (2) in overcoming friction of the

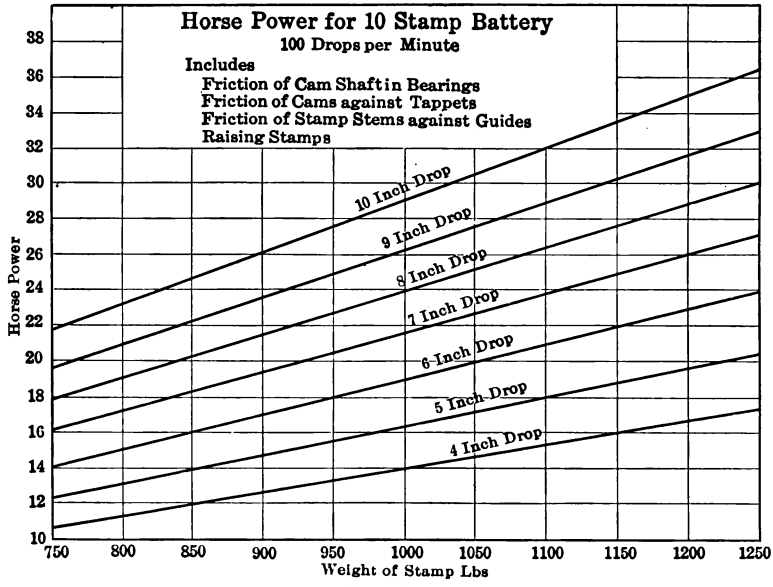

FIG. 35.

operating parts. The first item is easily calculated by figuring foot-pounds of work done and transposing into H. P., thus:

$$\frac{\text{Wt. of stamp} \times \text{lift in ft.} \times \text{number of drops}}{33{,}000} = \text{H. P.}$$

The second factor is somewhat indeterminate. After the power is transmitted to the cam-shaft, there remain in addition to the work already calculated, the following items; (1) the friction of the cam-shaft in its bearings, (2) the friction of the cam-face against the tappet, and (3) the friction of the stem within the guides.

Accurate data is lacking as to the total percentage of power thus wasted. With the increasing use of electric motors in individual units, it should not be difficult to obtain exact figures. It is probably between ten and twenty per cent of that required to raise the stamps.

The diagram in Fig. 35, taken from the Denver Engineering Works catalogue, shows approximately the power required for various drops with different weights of stamps.

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CHAPTER IV

STAMP MILL AMALGAMATION

General Principles. In standard stamp mill amalgamation two of the amalgamating methods outlined on page 9 are utilized; *viz.*, amalgamation by grinding-contact, and amalgamation by surface-contact.

That by grinding-contact is accomplished by feeding mercury, or as the millmen call it 'quick,' into the mortar with the ore, allowing the mercury to become broken up by the agitation and to come in contact with the newly liberated gold particles, thus serving the purpose of catching and retaining them within the mortar, or else of rendering them more readily caught by the outside plates acting on the surface-contact principle. The first method is generally referred to as 'inside amalgamation,' the second as 'outside amalgamation.' Outside amalgamation is used almost invariably; inside amalgamation is quite frequently omitted.

Inside Amalgamation. There are two ways of accomplishing amalgamation inside the battery; these we will designate as the 'loose-quick' method and the 'inside-plate' method.

In the first-named, mercury is fed to the battery in proportion to the amount of gold being handled and is expected to form amalgam. A certain portion of this amalgam will pass out through the screens; the balance will sink into the quiet spaces and between the dies, and there unite with other particles similarly deposited. If conditions are suitable amalgam may begin to form on the iron surfaces, and these may thus become centers for amalgam collection. This is not depended on, however, and the amalgam is looked for in the 'battery-sands' which collect between and around the dies. From these it is recovered on clean-up days when the mortar is emptied.

In the inside-plate method suitably shaped copper plates are placed within the mortar in the vicinity of the dies, serving thus to collect the mercury fed into the battery with the amalgam

formed by it. Practically all the pulp must come in contact with these amalgamated surfaces and in this case there is established within the mortar both grinding and surface-amalgamation contact. The use of inside-plates entails the use of wider and deeper mortars, and this for two reasons: first of all, for the accommodation of the plates and their wooden backing and, secondly, because it is useless to operate inside-plates with a shallow, narrow mortar, for the splash, under these conditions, is too violent. With the narrow mortar it is practically impossible for amalgam to accumulate on the plates, or if it does accumulate, there is serious danger that it will be scoured off with the least sign of over-feeding or other derangement within the battery.

This has been overcome in some cases by covering the inside plates with a coarse screen, which, in turn, means greater mortar width and consequently less capacity. It has therefore come to be recognized that where a high stamp-duty is required, *i.e.*, where capacity is of greater importance than close amalgamation, inside plates are undesirable. Also where the gold is coarse, so that both from weight and size, there is little chance of its escaping through the screen, inside plates are unnecessary. There can, however, be no question that with an ore in which the gold is difficult to amalgamate, better results can usually be obtained by the employment of inside amalgamation-plates, or 'coppers' as they are sometimes called. These plates are placed most commonly between the dies and the screen (Fig. 9, page 24). Sometimes both back and front plates are used (Fig. 11, page 25), in which case a projecting ledge is necessary to protect the back plates from abrasion by the coarse ore dropping from the feeder. End plates are rarely used. The inside plates are not usually open to inspection and in consequence much depends upon the millman's judgment and experience; if allowed to become too dry, amalgam is likely to break off, become pulverized, and pass out through the screen, where it may or may not be caught on the outside plates. If the amalgam becomes too soft, it may meet a similar fate, in which case, however, it is more likely to be caught on the outside plates.

Outside Amalgamation. An almost universal adjunct of the amalgamating stamp mill is the outside plate, or table as it is termed. This consists usually of a sheet of specially selected soft copper, in most cases silver-plated, with its surface covered with mercury or preferably amalgam; it is often by far the most suitable

means for the recovery of free gold in the pulp delivered by the stamps. These plates for a 5-stamp battery are usually from 5 to 6 ft. wide and from 8 to 24 ft. long.

In spite of many prejudices to the contrary, it is now generally conceded that silver plating of these copper-plates is a distinct advantage especially in starting a new mill. One ounce of silver per square foot is the usual thickness of plating, but two or even three ounces are sometimes specified. The area of the plate should, of course, be in proportion to the tonnage and value of the ore passing over it; from 3 to 4 sq. ft. per ton of ore per day will be found to accord closely with standard practice.

The grade of the plates is regulated by the character of the ore, the fineness of crushing, and the amount of water available. From $1\frac{1}{2}$ to 2 in. per ft. is about the average slope, although some authorities — notably the late W. J. Adams, a man of wide milling experience — insist that no plate should have less than $2\frac{1}{2}$ inches per foot slope.* Adams' argument is, that with less slope so much water is required that the film of pulp passing over the plates is too deep to allow the gold to come in contact with the plate, and also that it is possible to have too much mercury on the plate. The first argument loses weight when we consider the much slower progress of the pulp waves when the plate is comparatively flat, and the second point is weakened by the fact that with a good millman too soft a plate will not occur.

The proper appearance of the pulp when passing over the plate is shown by Fig. 36, the successive waves serving to turn the pulp over and over and thus to bring every particle in contact with the amalgamated surface. The progress of these waves should be as slow as possible, consistent with keeping the pulp moving and preventing the accumulation of 'sulphurets' (as the millmen call the sulphide minerals of iron, copper, etc.), magnetic iron, or other heavy particles on the plate. This means, that with a certain character of ore, the grade of the plate and the quantity of water used are mutually interdependent, and since, within the battery, a certain quantity of water is necessary to insure a satisfactory splash and discharge, the slope of the plate must evidently be such as to suit the pulp delivered by the stamps. Several drops or breaks in the length of the plate are often used; while formerly as much as 4 to 6 in. high, these are now seldom made to exceed

* Hints on Amalgamation, page 24.

1 in. in height. Their efficiency is usually proven by the accumulation of amalgam at the foot of the drop or fall.

The condition of the amalgamated surface of the outside plates, or 'apron-plates' as they are called, is the most important single factor in the satisfactory recovery of gold in stamp mill amalga-

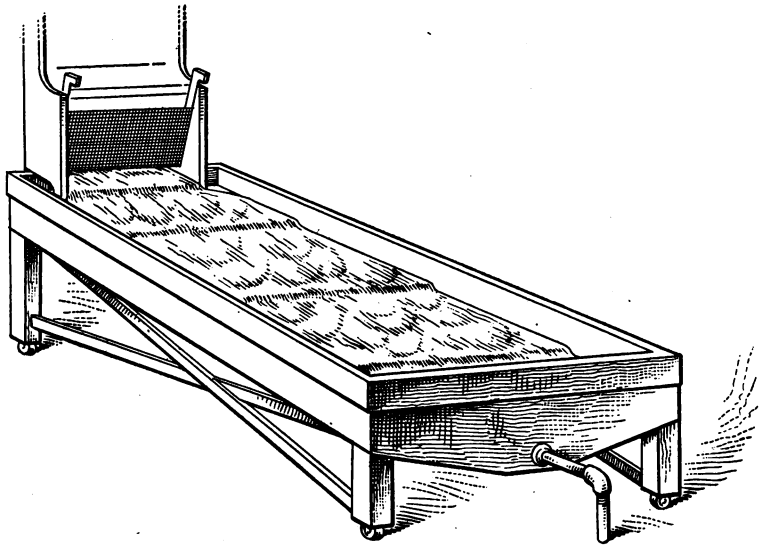

FIG. 36. Plate-Waves.

mation. If too hard the gold will not be caught; if too soft it is likely to be carried off by quicksilver flowing down the plate. The same thing is true if dry amalgam is discharged from the battery, in which case, as aptly pointed out by H. W. McFarren,* conditions are analogous to a flake of snow blown across a sheet of ice; if the ice is hard the flake will not catch, but the moment it strikes a soft or plastic spot it is at once arrested and held. The exact condition most suitable for any ore must be determined by experiment; coarse gold will usually be readily caught by a hard plate; finely divided gold, however, requires a soft plate. In general a white frosty appearance of the plate, and a consistency of the amalgam such that it will just yield to the pressure of the forefinger, is about right; but here, as indeed in almost every department of metallurgy, the ore in hand must be studied and suited.

* Min. & Sci. Press, Vol. 97, pages 814–816.

Where, as is usually the case, quicksilver is fed inside the battery, the plates are kept soft between dressings by this means. In fact the condition of the outside plates is usually the criterion for feeding mercury to the stamps. Where inside amalgamation is entirely omitted more frequent dressings are necessary. In some mills, operating under this condition, mercury is sprinkled on the plate while the pulp is going over it. This should only be permitted where the work is in the hands of an extremely expert amalgamator.

It must ever be kept in mind that amalgam is the best gold saver and that in consequence the plates must never be so thoroughly 'skinned' that a long period of inefficient gold saving must ensue before the plates are again in good shape. At the same time, when the amalgam begins to get thick on the plates it becomes difficult to prevent losses of amalgam by attrition. The medium here is determined mainly by the grade of the ore and the character of its gold, but it is found that, in the majority of cases, the amalgam which can be readily taken up with a rubber scraper should be removed every twenty-four hours and that the plates should be dressed every twelve hours.

In dressing the plates the stamps are hung up, or in some mills the pulp is diverted to another plate, and the plates to be operated on are washed clear of all adhering sand and dirt; then beginning at the bottom with a whisk broom or stiff paint brush, the plate is rubbed over thoroughly, loose pieces of amalgam are picked up, or bedded down, and where necessary fresh quicksilver is sprayed on the plate from a bottle the mouth of which is covered with canvas. Any spots which may show discoloration or coating are carefully cleaned, sometimes using a weak cyanide solution for the purpose. Some good millmen object to the use of cyanide and it will often be found that borax soap and 'elbow grease' are the only chemicals necessary to put a plate in good shape.

Amalgam is taken from the plate by the aid of a piece of rubber belting, or a plain rubber cut to a square edge. The amalgam is pushed from each side to form a ridge in the center, from this ridge it is collected in a small scoop, and then cleaned to await retorting.

Supplementary outside plates are often used in addition to the large expanse of apron-plate, and are usually placed close to the mortar. Lip-plates, secured to the projecting ledge of the mortar

below the screen (Fig. 9, page 24), are good amalgam catchers, as are also splash-plates which are placed in front of the screen on an inclined splash-board (Fig. 9, page 24). Splash-plates have the advantage that a large proportion of the pulp impinges against them with considerable violence. Sluice-plates, a survival of placer mining methods, are seldom seen nowadays; they are narrow plates in the bottom of the launder or sluice leading from the apron, and it is ridiculous to expect a narrow plate, deeply covered with fast-rushing pulp, to catch gold which has passed over the broad plate under so much more favorable conditions. Plates in the distributing boxes of shaking tables and vanners are occasionally quite efficient due to the agitation they undergo.

Mercury-traps at the foot of every apron plate are practically indispensable, as it is impossible to prevent quicksilver dripping from the lower end of the plate. Various forms of traps are used, that shown in Fig. 37 being the most common. The pulp is led through a pipe to the bottom of a well, from which it must rise to the overflow. Any mercury particles of appreciable size are arrested and remain in the trap, the bottom of which is provided with a drain or outlet. These mercury-traps act in some measure as a balance-wheel to steady the amalgamating conditions, and if the plates become dry or amalgam escapes, the trap often saves what would otherwise be lost. They should not be depended on for this purpose, however, and a large recovery of either quick-silver or amalgam from the traps is an indication of poor amalgamation on the plates or in the battery.

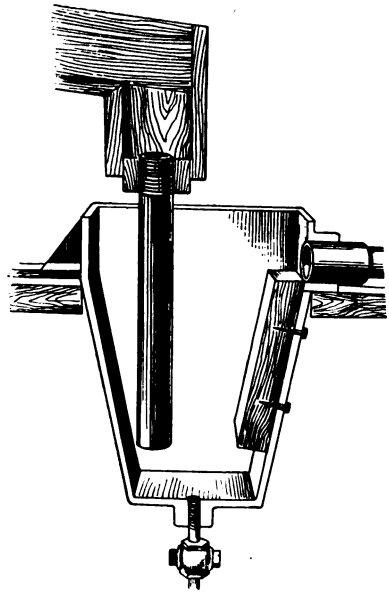

FIG. 37. Mercury Trap.

Position of Plates. In the majority of American mills the apron plates are placed immediately beneath the lip of the mortar, without the intervention of any distributing devices. In this

manner, the pulsating discharge from the battery screen is synchronous with the wave motion on the plate. When distributors are interposed between the mortar and its corresponding plate this is lost, and the wave motion must establish itself independently.

The apron plates should not be directly connected with the battery timbers as the vibration is thought to interfere with amalgamation.

In many of the South African mills where amalgamation follows the tube-mills, the plates are divorced entirely from the crushing machinery and placed in a separate building. It is a significant fact that under these circumstances the shaking copper-plate, which has never found much favor in this country, is extensively used and found to be more efficient than the stationary plate; especially should this be true with the thick, heavy pulp discharged by the tube-mills.

Plate Materials. Three materials for apron-plates are in common use: Raw-copper, silver-plated copper, and Muntz-metal. Silver-plates have been suggested as efficient; gold would be more so; neither are likely to become fashionable.

Raw-copper plates are cheaper than the silver-plated ones, but they are harder to get in shape, *i.e.*, to coat with amalgam, and in consequence, they do not become efficient gold catchers as soon as the silvered plates, and are more subject to discoloration and coating. They absorb more gold, but when once in good working order are certainly as good, and some authorities maintain even better gold savers than the silvered plates.

Silver-coated plates are more generally used than any other kind; they are, of course, expensive, but their promptness in getting into condition is usually sufficient compensation.

Muntz-metal is nothing more than a special brass (copper 60 per cent, zinc 40 per cent) formerly much used for plating ships' bottoms. It has long been popular in Australia and its use may perhaps become more general in the United States, especially where, as is not infrequently the case, cyanide solution takes the place of battery water. Under this condition Muntz-metal has been demonstrated to have a longer life than either plain or silvered copper.

Plate-Frames. The tables supporting the outside plates should preferably be of metal, if of wood the construction should be

thoroughly substantial. The grade should be adjustable and provision made for moving the plates away from the battery for repairs to the latter. The plates are secured to the table-top by means of cleats running along the sides. No screws should be put through the plate. Turning up the edges of the plate avoids any possibility of mercury leaking past the cleats.

Care of Quicksilver. The mercury used about the stamp mill must be kept free from impurities, if good amalgamation is desired. Only mercury which has been retorted is fit for use, and in some cases, this is not a sufficient guarantee of purity, as lead, zinc, and other readily volatile metals are distilled over with the mercury from the retort. In this event 'pickling,' usually with nitric acid, is resorted to, although both hydrochloric and sulphuric acid are used, the latter especially for removal of zinc. This pickling is accomplished by agitating the quicksilver with the acid until by its action the mercury is seen to be free from impurity. A simple mechanical test is sufficient in the majority of cases to detect harmful impurities. If the mercury is lively, and the globules in rolling about, retain their spherical shape, little need be feared, but if the particles are 'dead' and on rolling, drag behind them a tail, or are elongated, purification is necessary. Another test is to dissolve some of the suspected quicksilver in nitric acid; upon evaporating to dryness and igniting with sulphur, there should be no residue.

The supply of mercury for the mill should be issued in weighed amounts, and placed in a stout glass bottle or jar, the surface of the mercury being kept covered with dilute nitric acid or with a solution of potassium cyanide. From this receptacle the mercury to be fed to the battery is taken. Mechanical mercury-feeders have been devised, but they save no time in a gold-mill, and as the feeding of the quicksilver to the battery is a matter of judgment, based upon the condition of the lip or apron-plates, automatism is out of place.

The quantity of mercury used is, of course, proportional to the number of ounces of gold or silver amalgamated; usually from 2 to 4 oz. of mercury per ounce of gold is considered sufficient, but, as already suggested, much depends upon the character of the gold. The finer the particles the larger the amount of mercury required to form a suitable amalgam.

The loss of quicksilver is difficult to prevent. It is caused

by flouring and accelerated by sickening, the fine particles of mercury being carried out to the tailing pond. From 0.2 to 0.5 oz. per ton of ore are common figures for mercury loss.

A book should be kept for the mercury account in which the quantities issued and recovered are entered. Not only in the determination of cost are the figures of mercury consumption important, but in checking the work of the millmen. Excessive mercury consumption means one of two things, either the amal-

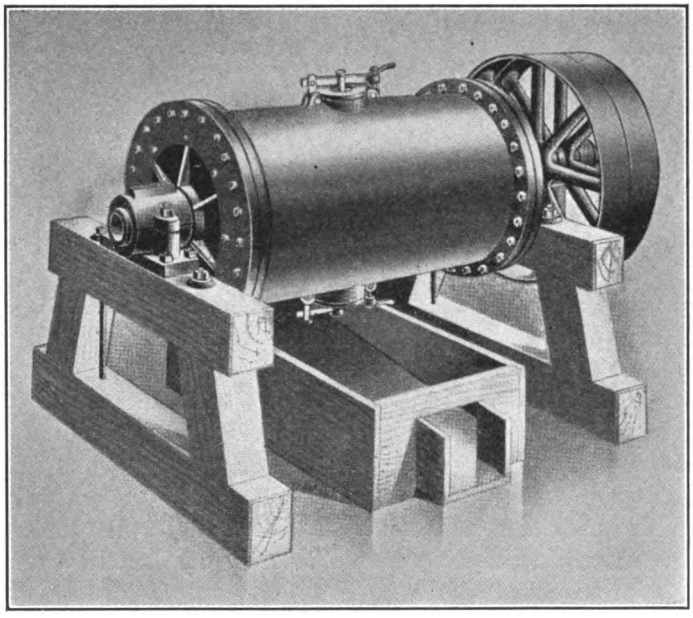

FIG. 38. Clean-up Barrel.

gamator is careless or he is dishonest. If the former, he may be spilling and slopping it about the mill or overfeeding the batteries, causing loss of gold. Otherwise the high consumption is due to gold, with its accompanying mercury, being stolen from the mill either by the millman or his accomplices.

Treatment of Amalgam. The soft, pasty amalgam, taken from the plates or elsewhere, is mixed with an excess of quicksilver and ground in a Wedgwood mortar, or a gold-pan, or in large mills in a clean-up barrel (Fig. 38), or pan (Fig. 39). After this treatment the impurities rise to the surface and are skimmed off and put aside

for future treatment. This grinding and skimming is repeated, adding in some cases hot water, until the mercury solution of amalgam shows a clean surface. It now remains to separate the amalgam from its excess mercury; this is done by pouring as much of it as can be conveniently handled into the center of a cloth previously wetted and so placed over a vessel as to form a suitable depression. The ends of the cloth are then gathered up and the mercury is squeezed through the pores of the cloth, leaving behind a solid lump of amalgam. The lump is rolled about over the cloth

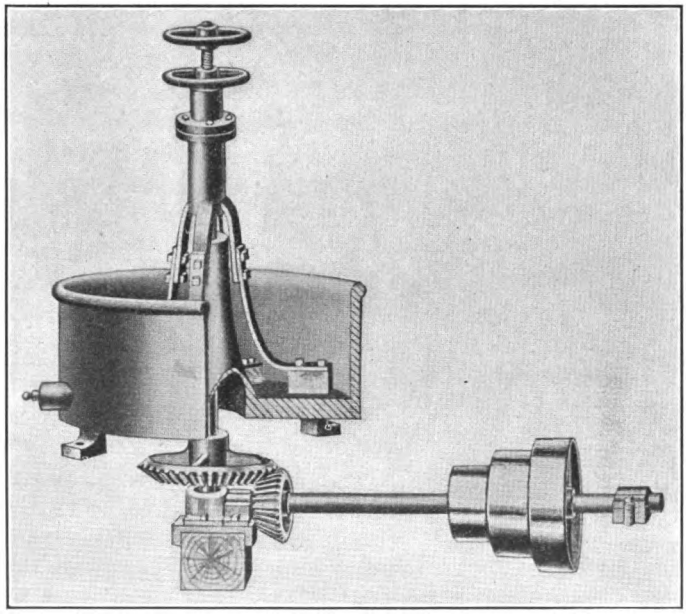

FIG. 39. Clean-up Pan.

to pick up any small, detached particles adhering to the folds, and is then placed in the safe to await retorting. The process is repeated until the entire batch is squeezed.

The quantity of mercury left in the squeezed amalgam is dependent upon its temperature, the extent to which it has been squeezed and worked, and the character of the gold. From one-third to one-fourth bullion is expected from any given weight of hard amalgam. Where large quantities of amalgam are to be squeezed the work is done by the plunger of an hydraulic or other power press (Fig. 40). The mercury thus recovered from the

amalgam solution always carries an appreciable quantity of gold, the amount depending upon the size of the gold particles, the kind of medium used for the squeezing, and the temperature of the water. This gold (or silver), far from being a detriment to the quicksilver, is a positive advantage, and provided the mercury contains no other impurity it may be used again in the battery. It should be tested however. Canvas or the so-called 'outing flannel' of a stout quality is generally employed for the filtering medium. Almost all writers on the subject mention chamois skin, but it is seldom used, and my own experience has been that it rots with the repeated wetting and drying and soon becomes too weak to stand the pressure.

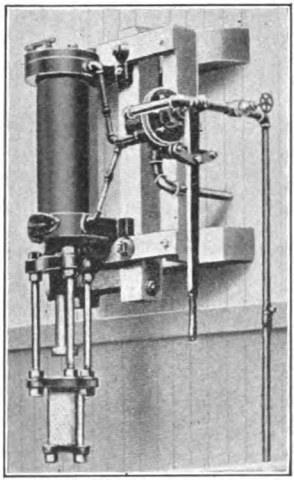

FIG. 40. Amalgam Press.

The dross and skimmings are allowed to accumulate along with other odds and ends from about the mill, until a sufficient quantity is on hand to justify cleaning and retorting separately. The quicksilver recovered from such material needs special handling before being returned to the mill supply.

Aids to Amalgamation. The number of nostrums, 'dopes,' etc., which have been proposed as infallible aids to the recovery of gold by amalgamation is endless; few of them are worth the space required to discuss them.

Sodium-amalgam, first proposed according to Rose * by Sir Robert Crookes, seems to be efficacious in preventing the accumulation of a layer of metallic oxides on the surface of the plates. This preventive action is probably due to the hydrogen released by the decomposition of the water in contact with the metallic sodium. The resulting caustic soda is also of use in 'killing' the grease which is a serious hindrance to amalgamation. The majority of millmen do not care to use sodium-amalgam, owing to the great avidity with which it amalgamates with iron, for when too much is used the inside of the battery, as well as the shoes, dies, and screens speedily become coated with amalgam. Where

* Metallurgy of Gold, 3rd Ed., page 144.

much difficulty is encountered in keeping the plates bright, a small amount may prove beneficial, and for this reason the method of preparing it will be given. A 20-gram assay crucible is half filled with mercury and placed in a support over a Bunsen burner or on a laboratory hot-plate. When thoroughly heated to about the boiling point of water, cover the crucible with a scorifier or crucible-cover, then, cautiously raising the cover, drop in a piece of metallic sodium about the size of a pea, and at once replace the cover. When the pyrotechnic display has ceased, more sodium is cautiously added in the same way, until the amalgam is in such condition that it will just begin to adhere to a freshly filed iron surface. When this point has been reached the amalgam is cooled, and placed in a tightly stoppered bottle; it may be substituted for plain mercury both inside and outside the battery.

Cadmium- and zinc-amalgams have both been investigated by C. H. Aaron,* who found cadmium-amalgam especially to be remarkably free from tarnish under conditions where mercury-coated copper-plates were quickly discolored. Both of these amalgams should be investigated more fully.† It is possible that the claimed superiority of Muntz-metal plates may be due in part to the zinc-amalgam which will naturally be formed on their surfaces.

Electrolytic Amalgamation has, from time to time, been heralded as another panacea for all the amalgamator's ills.‡ The results would seem, in so far as they are beneficial, to resemble those produced by sodium-amalgam, the electrolysis liberating hydrogen by decomposing water and depositing metallic sodium, when, as is often recommended, salt-water is fed to the batteries. As the cyanider has found to his cost, however, electrolysis is likely to quicken the action of those reagents which are valuable to us because of their selective tendencies, with the result that they become more promiscuous, taking to their bosom substances we do not wish to recover by their aid. Practically, electrolytic amalgamation has a tendency to amalgamate everything in sight and

* Eng. & Min. Jour., Vol. 48, page 118.

† Recent investigations give following results: Amalgams in order of ability to resist coating by $FeSO_4$ and $MnSO_4$: Cd, Pb, Sn, Au, Ag, Na, Zn and Hg. Amalgams in order of their ability to save gold in presence of sulphates: Sn, Pb, Na, Ag, Zn, Cd, Hg. (Unpublished thesis of R. Keffer, State College of Washington.)

‡ Min. Sci., Vol. 60, pages 30-32.

hence has never found general favor. As in the case of sodium-amalgam, there are probably special cases where an extremely weak galvanic action will prove beneficial in keeping the plates bright and free from tarnish. This may easily be arranged by placing a few iron strips lengthwise on the amalgamated copper plate.

The Alaska-Treadwell company reports an increased saving by amalgamation when using sea-water in the stamp batteries. That this is probably due to electrolytic action is suggested by the further statement that the corrosion of the mortars is so great that the net result is undesirable financially.

Chemical Aids. Lime, caustic soda, caustic potash, wood ashes, and other substances of like character are fed to the battery in cases where the ore, due to partial oxidation, contains free acid. Their use under this condition is logical and to be commended.

Potassium cyanide is almost universally condemned by writers on stamp milling, and almost as generally believed in by skilled amalgamators. As a plate dressing solution it is more commonly used than any other chemical. It is sufficiently alkaline to remove any film of grease, is a strong reducing agent, and dissolves most metals and metallic oxides. It has a tendency, however, the reason for which has never been explained, to harden the amalgam by causing the excess of mercury to exude from the plate and run down to the lower end, thus leaving the amalgam dry and hard. In moderation and in weak solution its use is justified. The fact, however, must not be lost sight of that both gold and gold-amalgam * are soluble in cyanide solutions. Ammonium chloride has many of the properties of cyanide in this connection with the added advantage of not being a solvent of gold. It is frequently of decided benefit with discolored plates. Extremely dilute solutions of hydrochloric and sulphuric acid are at times of benefit.

Ferro and ferricyanide of potassium as a mixture for dressing plates is highly recommended by W. J. Adams,† the compound being made by adding 1 part of a saturated solution of ferrous sulphate to 4 parts of a 2.5 per cent cyanide solution. This is a good example of the prescriptions so sacredly guarded by the older generation of millmen. The fact that many of these compounds

* According to Roskelley 4.65 oz. of amalgam lost 0.025 oz. in 12 days in 3 per cent KCy solution.

† Hints on Amalgamation.

were known only to a few in nowise enhanced their value, for a more general knowledge of them would have led to more severe trials and the abandonment of all but the best.

The greatest enemies to successful amalgamation either inside or outside the battery are the films either preëxisting on the gold in the ore, or created artificially on the gold, on the mercury globules, or on the plate. These films, often of infinitesimal thickness, are sufficient to prevent contact without which, of course, no amalgamation can be expected. Grease and various metallic oxides are the main film-forming substances; the former is, of course, saponified by the alkalies. The metallic oxides may be dissolved by the alkali, but are more likely to be removed by a solvent such as cyanide or a mineral acid. Since cyanide is decidedly alkaline and at the same time a good solvent for most of the oxide films, its popularity as a plate dressing seems logical enough.

Increased Temperature has been claimed to be an aid to amalgamation. T. T. Read, whose work on amalgams has already been mentioned (page 8), concludes that increased amalgamation temperature, which, of course, is accomplished by increasing the temperature of the battery water, has the following effects: It increases the solubility of harmful salts existing in the ore, and hence the precipitation of base metals into the amalgam. It decreases the viscosity and surface tension of the mercury and flouring is thereby increased. The power of the mercury to hold the gold particles to the plate is also diminished. Increase of temperature is beneficial in that quicksilver is more lively when warm, catches the gold more readily, and separated particles are more easily made to coalesce. Read says: "A comparatively low temperature is better where the influence of soluble salts is to be considered, but when this may be neglected as high a temperature as can economically be maintained *without variation* * is most favorable to successful amalgamation." The gist of the above quotation is in the italicized words "without variation." In the vast majority of cases any temperature within reasonable limits will give satisfactory results, provided there is but little fluctuation.

Cleaning Plates. The daily clean-up, by means of rubber scrapers, fails in the majority of cases to remove completely the amalgam adhering to the plates. This result is cumulative and

* The italics are mine, F. A. T.

eventually in every mill a point is reached where more drastic measures must be used. While it is universally admitted that amalgam is the best gold catcher, it is not expedient from any point of view to allow a large amount of gold to remain on the plates indefinitely. In a custom mill, of course, practically all amalgam must be removed from the plates at the end of every run.

Hard-wood scrapers are frequently substituted for the rubber ones, and will remove more amalgam. Steel scrapers are made by forging each end of an old file to a chisel edge, one end being turned down to a hoe-shape. In Gilpin county a better form of scraper is employed. This consists of a piece of steel 12 by 3 by $\frac{1}{8}$ in. thick, turned down for about $\frac{3}{4}$ in. at one end. The edge of the turned end is ground perfectly square, and the square edge thus made is depended upon for the scraping effect. In this form the corner of the scraper is not so likely to dig into the plate and make an objectionable groove.

The steaming of plates for the removal of hard amalgam is increasing in favor, and is a much cleaner and neater method than the old system of taking the plates from the table and placing them over a charcoal fire, after which they were seldom fit for anything but the scrap heap.

Rickard * in describing the operation of cleaning the plates at the Alaska-Treadwell Mill says:

"Their surface is first washed with a jet of water to remove particles of ore. A wooden cover is laid over the copper plates, resting on the cleats that fasten the plates to the table; this leaves a $1\frac{1}{2}$-in. space between the cover and plates. All openings along the sides and at the top and bottom are closed with strips of burlap; a hose is inserted at a point one-third the way from the head of the plate and live steam is admitted at a pressure of 36 to 65 lb. The steam is allowed to act for 20 min. and is then turned off. The wooden cover is removed and the softened amalgam is scraped off with chisels. The amalgam has become as soft as Swiss cheese and about as brittle. The mercury is partly volatilized by the heat, and the men are careful to stand back when the cover is raised. Warning is given by the fact that the brass sockets of the electric lights hanging over the tables become amalgamated. In cleaning the tables the men wear a wet sponge at their nostrils. When the amalgam has been scraped off, the plate is washed with a weak cyanide solution; it is then dressed with fresh mercury, using whisk-brooms to spread the globules of mercury uniformly."

* Min. & Sci. Press, Vol. 98, page 861.

All such methods, however, should be regarded in much the same way that the general public regards a surgical operation — as a thing of the last resort.

Eventually all amalgamating-plates that have been long in use find their way to the copper smelter for extraction of absorbed gold and silver.

Battery Clean-up. Wherever mercury is fed into the battery, a periodical cleaning out of the mortar must be made to remove accumulated amalgam. If coarse gold is present in the ore part of it will accumulate inside the mortar whether mercury is used or not, and this alone would make a periodical clean-up necessary.

The interval for this operation is dependent entirely upon the richness of the ore. In low-grade mills the battery clean-up is made once or twice a month. In contrast with this I have had to make a clean-up every day when milling exceptionally high-grade material.

The method of cleaning-up is usually as follows: When ready to close down the mill, the feeders are shut off and the battery is allowed to continue running. This is continued as long as is consistent with the safety of the shoes and dies; the water is then turned off and the stamps are hung up on the latch-fingers. The screen-frames are taken out, carefully washed over the apron-plates, and examined for adhering amalgam, which, if present, is removed and placed in a granite-ware kettle. If inside plates are used they are removed to a place of safety and their amalgam removed by scraping. If the plate-frames are portable they are moved from the front of the mortar, if not the workmen stand or squat on planks placed across the plate-frame, and the remaining battery-sand is taken from between the dies and put aside for future treatment. A sharp look-out is kept all the while for loose lumps and balls of amalgam. The dies are worked loose with a crowbar (not usually an easy operation), cleaned of adhering amalgam and set aside. The bottom of the mortar is carefully scraped, crevices and corners are examined, and when all is perfectly clean, a layer of tailing is spread on the mortar-bottom, and the dies, if not worn out, are set in place again and the tappets are re-adjusted. If the dies have worn irregularly they are put back in the mortar in a reversed position to equalize the abrasion. The inside-plates having been cleaned of their amalgam are dressed with quicksilver and set in place. Screens and screen-frames are renewed if neces-

sary, keyed in position and the mill is again ready to begin crushing ore. One unit, usually of ten stamps, is thus cleaned up at a time and is started running before the next one is stopped. The time consumed is from 45 min. to one and a half hours per five-stamp battery. A small trolley-track suspended from the cam-shaft floor, upon which a bucket may be run in front of the batteries, is a convenience in moving the material to be handled.

Cleaning Battery-Sand. The removal of amalgam and coarse gold from the battery sand is accomplished in various ways. In small plants the material is usually panned or run through a sluice-box provided with riffles, the tailing from the process being returned to the battery. In larger mills three methods are in use:

(1) Returning the sand from the different batteries to one mortar, which is cleaned-up last of all.

(2) Charging all battery-sand into a clean-up barrel, rotating about a horizontal axis. (Fig. 38.) In this the battery residue is ground for 6 to 12 hours, with an excess of mercury. Cast-iron balls are used to loosen the amalgam from all particles to which it has been adhering. It is collected by the excess mercury and is drawn out and taken to the clean-up room and panned to free it from gangue matter. In some mills the material from the clean-up barrel is discharged into a riffled sluice, the riffles being depended upon to catch the heavy particles. Mechanically operated pans or bateas take the place of the gold pan in some large mills.

(3) Grinding the material collected from the mortars in a small pan, on the principle of that used in the silver mills (Fig. 39). This last, however, is better suited for cleaning large quantities of amalgam previously freed from all but the finest impurities. Battery-sand is likely to contain pieces of iron, such as pick-points, ends of drill-steel, particles of shoes or of dies which would interfere with grinding it. When the grinding-pan is used a gentle stream of water is kept flowing into the pan. This serves to carry away the finely ground ore or slime leaving only the heavy sulphides, iron and mercury to be dealt with by hand.

The final collection of amalgam is treated, as previously described, with a large excess of quicksilver to free it from impurities and is then squeezed through cloth or by an amalgam-press, Fig. 40.

Cleaning-up of odds-and-ends. Nothing which is used about the clean-up room or around the batteries, where amalgamation

is practised, should be destroyed. Whisk-brooms, plate-rubbers, old screen-frames, chuck-blocks, amalgam-cloths, chips from inside the battery, dross and scum from amalgam, and a thousand other little scraps, should be carefully kept, and if combustible, burned and the ashes either returned to the battery or placed in the clean-up pan or barrel. Iron scraps should be thoroughly rusted with the aid of salt or sal-ammoniac to loosen any adhering films of amalgam, and may then be similarly charged into a barrel although preferably made a separate charge.

Clean-up Room. The clean-up room need not be large. It should be kept securely locked whether in use or not. The windows should be inside the mill and heavily barred, or should be omitted altogether, provision being made for ventilation through the roof, and light being furnished by electricity. The fittings should consist of a deep sink or two for panning, and a flat-top table each side of the sink and sloping slightly toward it. Concrete is the best substance for the sink and table-tops and the floor should be of the same material. The amalgam-press, if one is used, should be placed here. A small safe for temporary disposal of amalgam is useful. Whether the retorting and melting is done in this room or not is a matter of individual preference; usually where a stamp mill is operated with a cyanide plant in conjunction, retorting, melting, and refining are done in a special building.

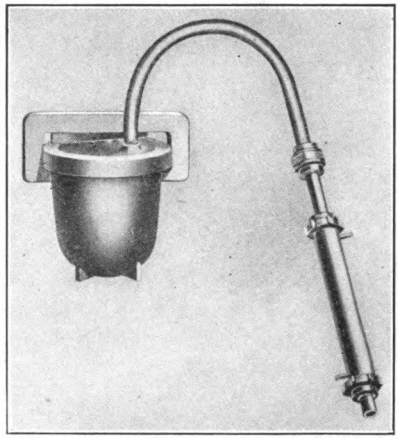

FIG. 41. Pot-Retort and Condenser.

Retorting. The hard amalgam, whether squeezed by the press or by hand, must be heated to free it from mercury before it is fit to be melted into bullion. To do this it is handled in some form of retort, the distilled mercury being condensed and recovered.

In small mills the pot-shaped retort (Fig. 41) is used. In large mills a retort in the form of a cast-iron muffle is built, and the amalgam is placed therein on small trays or pans. When using

the pot-retort, the amalgam should not more than two-thirds fill the vessel, which should be chalked inside, or lined with tissue paper. The cover secured by a key and clamp is made tight with a lute of salt and bone-ash, or of fine slime. Heat is applied gradually and water is turned into the jacket on the condensing pipe. As soon as mercury begins to distill, the heat should be checked slightly and should then be regulated according to the

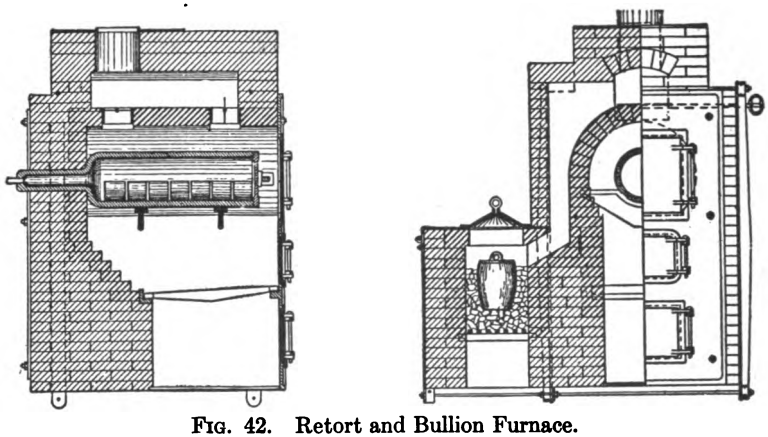

FIG. 42. Retort and Bullion Furnace.

flow of mercury. When no more quicksilver is given off, the heat should be raised to bright red to expel the last traces, and the retort allowed to cool. The spongy mass, called by millmen the *re*-tort, is turned out and is ready for melting. It always retains a small amount of mercury, but if reasonably free from sulphides and other impurities, will be a characteristic golden-yellow. The built-in retorts (Fig. 42) have, of course, a stationary fire box. The smaller retorts may be heated in any convenient manner. An ordinary gasoline assay furnace serves admirably, and special retort furnaces (Fig. 43), designed for the use of crude oil or gasoline, may be bought. There are two sources of danger in retorting;

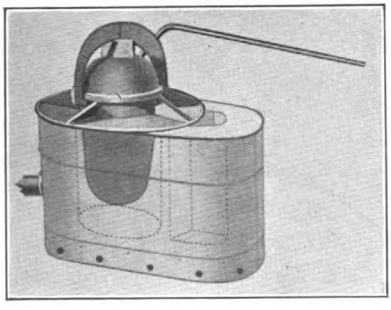

FIG. 43. Gasoline Retorting-Furnace.

the pipe may become plugged with particles of amalgam, due to the violent boiling which takes place when the retort is heated too rapidly; or the end of the pipe may be carelessly allowed to dip under water, with the result that should the heat be decreased, the condensation of the vapor will draw water into the retort. Either of these accidents is almost certain to result in an explosion, scattering the valuable contents of the retort in all directions and exposing the workman to danger of salivation by mercury vapor.

Melting. The product of the retorting operation is melted in graphite crucibles, usually in an ordinary coal or coke-fired wind-furnace, although crude-oil or gasoline is infinitely superior for the purpose, on account of the readiness with which the heat may be controlled, as well as the efficiency and convenience in their use.

Mills seldom do any extensive refining of their gold; a little borax or occasionally a mixture of borax and niter is added to the crucible by means of a small metal spout. While the elaborate precautions described by the older authorities for the preparation and annealing of the graphite crucibles are scarcely necessary now-a-days,* it is well to store the crucibles in a warm place, such as the top of a furnace, to ensure complete elimination of moisture; they should also be tested by fairly slow heating in the furnace before the bullion is placed in them.

Pouring-molds are usually rectangular in shape, and before the molten gold is poured into them they should be given a coating of graphite on the inside and placed on the furnace to heat. A piece of tallow or paraffine candle is allowed to melt in the mold, till the bottom is covered about $\frac{1}{4}$ in. deep. This should be smoking slightly when the gold is poured. If a large amount of slag is formed in the crucible it is skimmed off; otherwise it is poured with the gold, and subsequently knocked off with a hammer, or detached by throwing the bar, while hot, into water. If the bar is not smooth, either the mold was too cold, there was not enough grease, or too much slag was poured with it. If desired it may be remelted without appreciable loss. A high polish may be given to a bar by dropping it into dilute sulphuric acid before it has time to cool. In small melts the crucibles are handled by hand-tongs. With larger quantities a pair of scissor-tongs with a small crane is necessary.

* I have, as an experiment, actually taken a new crucible, filled it two-thirds full of retorted gold, and placed it in the full heat of a crude-oil burner without the least injury to the melting pot.

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CHAPTER V

VARIATIONS IN PRACTICE

For the purpose of discussing the modifications and applications of the stamp mill, the various practices will be classified as follows:

(1) *Gilpin County Practice, using light stamps, high discharge, and high drop with slow speed, relying largely on inside amalgamation with the aid of inside plates.*

(2) *Original California Practice, using a medium weight stamp, moderately high discharge, inside plates, short drop, and high speed.*

(3) *Modified California Practice, using a heavy stamp, low discharge, short drop, and high speed, feeding quicksilver into the battery but using no inside plates.*

(4) *South African Practice, using the stamp mill primarily as a crusher, with extremely heavy stamps, short drop, high speed, and low discharge and omitting inside amalgamation altogether.*

Table II, giving important features of typical mills in each class, will exemplify the points of difference.

TABLE II

Class No.	Wt. of stamp	Height of drop	No. of drops per min.	Height of discharge	Area of inside plates	Stamp duty	Screen used	Proportion of amalgam recovered in battery
	lb.	in.		in.	sq. ft.	tons	mesh	
1	500 to 600	16 to 18	25 to 30	11 to 16	$6\frac{3}{4}$	1 to $\frac{1}{2}$	50	$\frac{2}{3}$
2	850 to 950	6 to 8	90 to 100	7 to 8	$1\frac{1}{2}$ to 2	2.5 to 4	30 to 50	$\frac{1}{2}$ to $\frac{2}{3}$
3	850 to 1200	6 to 8	90 to 110	2 to 4	none	3 to 9	20 to 40	$\frac{1}{4}$ to $\frac{1}{2}$
4	1200 to 2000	4 to 7	90 to 115	2 to 4	none	7 to 20	4 to 10	none

(1) GILPIN COUNTY PRACTICE

Gilpin County practice developed at a place where milling was first done with the California stamp mill. Gradually the ores became more complex and the mills, failing to make the extraction, gave way for a time to the smelters. Eventually, however, the

millman triumphed, by adapting and modifying his amalgamation practice to suit the ore. He found that a roomy mortar, with plenty of inside-plate surface, and fine-grinding, assisted by deep discharge and long contact of the pulp with the plates and the quicksilver in the mortar, would give a satisfactory extraction from an ore, in which the gold was intimately associated with pyrite, and so fine as to be rarely visible to the naked eye. In short, the Gilpin County millman is successfully treating by amalgamation, aided by the crudest kind of concentration,* an ore which practically every other millman in the world would declare could not be handled by stamp milling. The capacity per stamp is far lower than elsewhere, but the cost per ton bears favorable comparison with costs in other plants and the extractions are good. For various reasons, however, the method of milling developed in the 'Little Kingdom' of Colorado has not been extensively used, away from its birthplace. The following experience in a small mill in the Southwest is of interest in this connection: The mill was equipped with 850-lb. stamps, making 104 to 106 five-inch drops per minute. The discharge was 4 in., no inside plates were used, and loose quicksilver was fed to the battery. Pending the installation of a cyanide plant, one battery was made as nearly as possible a Gilpin County mill. The discharge was made 9 in., the drop 10 in., and the speed was reduced to 50 drops per minute. Inside-plates were arranged in both back and front of the mortar. Under the first condition we were saving 40 per cent by amalgamation and our concentrate assayed $200; using the 'Gilpin County Mill,' our amalgamation extraction went up to 65 per cent and our concentrate dropped to $75 per ton. This illustrates the essential feature of the Gilpin County practice, *viz.*, that it saves gold by amalgamation which would usually go to the concentrator or to some other form of treatment. And, therefore, we may safely say, that the Gilpin County stamp mill, with its apparently pettifogging ways, its small tonnage, and its conservatism, is without question — outside possibly of the arrastra — the best amalgamator in the world.

(2 and 3) THE CALIFORNIA PRACTICES

The California type of mill antedates the Gilpin County Mill and is more nearly representative of the usual, standard practice

* The Gilpin County bumper can only be called a concentrator by courtesy.

throughout the world to-day than any other style of milling. In the design of the mortar, the aim has constantly been to harmonize capacity with inside amalgamation. A narrow, deep mortar has been found to combine these two essentials (see Fig. 9, page 24). The front plate is placed above the level of the dies to avoid the scouring action at that point.

The California milling fraternity is, however, far from being united as to this being the better practice, and that which I have designated the 'modified California' practice has many adherents. Faster crushing is possible where inside plates can be omitted, owing to the decreased height of discharge. There are good reasons, however, for feeding quicksilver into the battery whether inside plates are used or not. In the first place, the particles of gold as they are liberated, brightly burnished from their quartz matrix, are in excellent condition for instant amalgamation. Once amalgamated they become pasty and malleable and are less likely to be broken up by the blow of the stamp; furthermore, should they then be broken, they are more likely to be caught by the plate than the corresponding particles of gold would be if unamalgamated. There is further a strong tendency for particles of amalgam within the mortar to coalesce, and so protect each other from abrasion and from being carried out with the swash of the pulp. It is, moreover, an axiom in milling, that the gold should be saved as early in the process as possible. If one saw a $50 gold nugget in the ore bin, he would be foolish to let it go into the mill and take a chance of its being broken up and in part floured. Similarly that which can be saved within the battery should not be allowed to run the risk of loss on the plates outside and so on. As to whether inside plates should be used or not depends upon the ore. With high-grade ores they undoubtedly should be used; with low-grade ores, it would seem less desirable and yet we have the examples of the Homestake and the Alaska-Treadwell mills both using inside plates with the lowest-grade gold ores that are being treated at the present time.

(4) SOUTH AFRICAN PRACTICE

South African practice has come, in recent years, to an entire omission of inside amalgamation. The stamp mill thus becomes a crusher of rock, pure and simple, and it is at once on debatable ground as to its supremacy over all other machines. As a com-

bined amalgamator and crushing machine the stamp is without an equal. Remove the amalgamating feature and the superiority of the stamp over other devices is a matter of doubt. However, at the present time in South Africa, heavy stamps are extremely popular. The tendency is to make the stamps crush only to a coarse product, from $\frac{1}{8}$ to $\frac{1}{20}$ in. diameter, relying on the tube-mills or other devices to reduce the ore to the sizes deemed essential for cyanide treatment. Here, amalgamation in many cases follows not the stamps but the tube-mills.

Each of these practices shades and blends into the other, so that whereas a particular mill may not fall into any one of the four classes specified, at the same time it is likely to approximate closely to one or the other.

INDIVIDUAL STAMPS

All of the preceding discussions have presumed that the batteries under consideration were of the five-stamp type. In the last few years important installations of one- and two-stamp mills have been made and much valuable data has been collected on the subject. The advantages claimed for stamps operating in individual mortars are: (1) Greater screen area and consequently greater tonnage. (2) Absence of interference by waves from other stamps and consequently more efficient crushing, resulting in less power per ton stamped. (3) Less slime produced. These mills, however, have not fully justified anticipations. They take more labor per ton than the five-stamp battery, for screens, feeders, and other apparatus are vastly increased in number. The swash within the individual mortars is so great that inside amalgamation by plates or otherwise is out of the question. The tonnage per stamp is undoubtedly greater and the power consumption per ton is less than for the same weight of stamp in a five-stamp battery. Where the conditions discussed under 'South African practice' are

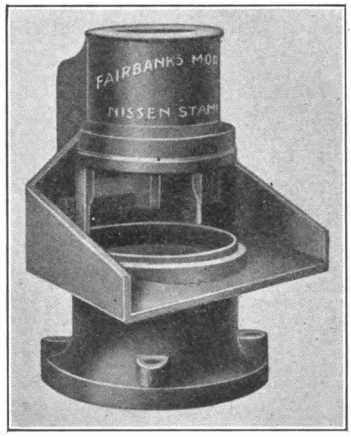

FIG. 44. Nissen Mortar.

to be met it would seem that single-unit stamp mills might have a field.* The Nissen stamp (Fig. 44) and the three-stamp individual-mortar mill, of the Joshua Hendy Co. (Fig. 45), are the best known representatives of this class. Two-stamp batteries (Fig. 46) are sold but seem to have little to commend them; they have

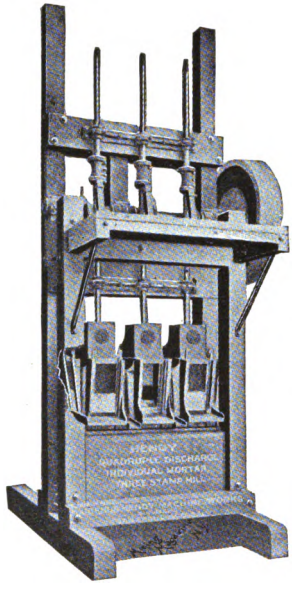

FIG. 45. Three-stamp Mill Individual Mortars.

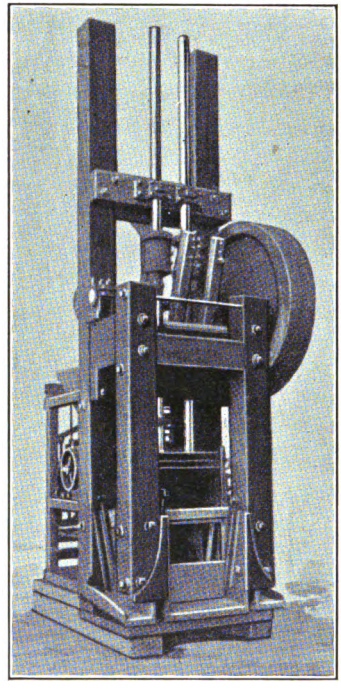

FIG. 46. Two-stamp Triple Discharge Mill.

neither the capacity of the unit-stamp, nor the amalgamating advantages of the five-stamp, mill.

STAMP MILLING COSTS

Stamp milling costs are made up of the following items: (1) power, (2) labor, and (3) repairs and supplies.

(1) **Power.** This item is, of course, dependent upon the kind of prime-mover used. It may be a steam-engine, an oil or other

* Recent reports of Nissen stamp performances in South Africa appear to justify this opinion.

internal combustion engine, or it may be a water-motor. Any of these may drive the mill direct or may serve to operate an electric generator. The cost per horse-power per annum will vary with electric or steam power from $40 under unusually favorable circumstances to as much as $150 in small plants using expensive fuel. Take as an example a 1050-lb. stamp dropping 6 in., 100 times per minute. This means a consumption of 2 h-p. (see Fig. 35, page 50). Assuming a stamp-duty of 3 tons we get 0.66 h-p. per ton per 24 hours. Figuring 300 working days to the year and $60 per annum as a good average, our power cost is 20 cents per h-p. per day or 13⅓ cents per ton. Similar calculations will enable the power cost to be readily determined for any given data.

(2) **Labor** is not so easily calculated since it is often difficult to segregate the service rendered in different parts of the plant.

Taking a 60-stamp mill, one amalgamator, and half of the time of one helper per shift (the rest of the helper's time being devoted to the rock-breaker, etc.) with half the time of the mill foreman will cover the actual stamp mill labor. If, as is common, 8-hour shifts are used, the cost is, of course, more than where two shifts only are necessary.

	12-hr. Shifts	8-hr. Shifts
½ Foreman @ $4.00	$ 2.00	$2.00
1 Amalgamator per shift @ $3.50	7.00	10.50
½ Helper per shift @ $2.75	2.75	4.12½
	$11.75	$16.62½

With our stamp duty as before this gives 6½ cents for labor where two shifts are employed or 9¼ cents with three shifts. As the size of the mill decreases the cost for labor will of necessity increase. On the other hand, increasing the size of the plant does not decrease the labor cost but rather tends to increase it until another point of maximum economy is reached. The lowest labor cost for milling is probably that of the Alaska-Treadwell Co., 2.9 cents per ton.

(3) **Repairs and Supplies.** Wear and tear on the stamp parts, lubricants, water, and quicksilver are the important items under this head. Shoes and dies — the parts subjected to abrasion — are, of course, the heaviest items of expense. Rickard,* writing fifteen years ago, summarizing practice in Colorado, California, and

* Stamp Milling of Gold Ores, 1st Ed., pages 206–209.

Australia, gives the wear of the shoes as from 0.24 to 1.33 lb. per ton of ore crushed; of the dies from 0.19 to 0.5 lb. per ton. The costs correspondingly range from 2.02 to 7.64 cents for shoes, and from 0.7 to 5.5 cents for dies. The combined cost for both shoes and dies he gives as ranging roughly from 4 to 13 cents per ton. The minimum cost for dies in the examples cited by Rickard occur with wrought-iron, chrome-steel giving the lowest cost for shoes. The Alaska-Treadwell mills, using Koppel-steel shoes and locally-made cast-iron dies, reports wear per ton as 0.27 and 0.16 lb. respectively. The cost of material for shoes and dies will vary from 3 cents per lb. for cast-iron, under exceptionally favorable circumstances, to 6 or 7 cents for chrome, or other good quality steel at an average western railroad point. Richards * gives as the average cost for shoes and dies of various kinds in 14 mills, 5.03 cents per ton. This is probably high. The wear and tear in other parts of the stamp mill, of course, vary greatly with operating conditions. Including everything about the battery not otherwise specified, it will probably run from 2 to 5 cents per ton in a well-operated plant. Screen-cost will run from a maximum of 3 cents per ton, under the severe conditions of fine screen and low discharge, to as low as 0.15 cent (estimated screen-cost at Alaska-Treadwell mills), thus making the total for wear and tear on stamp parts average as follows:

Shoes and dies	2 to 13¢ per ton
Other stamp parts	2 to 5¢ per ton
Screens	0.2 to 3¢ per ton
	4.2 to 21¢ per ton

The variation in the total would probably not be as great as this, since some of the factors will tend to balance one another.

Lubricants, water, and quicksilver are likely to be the most variable items in the operating cost. The first, however, is so small as to be insignificant. The water cost may vary from practically nothing to as much as 10 cents per ton of ore in arid regions even when every precaution for saving and re-use is taken. Quicksilver consumption per ton of ore is also a variable factor but will average between 0.1 and 1 oz. per ton of ore, or from 0.3 to 3 cents per ton.

* Ore Dressing, Vol. 1, page 206.

Summarizing the actual running expenses under the usual range of conditions, and omitting such items of general expense as superintendence and lighting, depreciation and interest on the money invested in the plant, we have as follows:

Item	Cost in cents per ton milled Maximum	Minimum
Power	25	5
Labor	20	3
Stamp parts, shoes, dies, and screens	21	4½
Water	10	0
Quicksilver and lubricants	4	1
Totals	80	13½

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CHAPTER VI

OTHER MILLS AND GRINDERS

The other mills in use may be divided into:
(1) *Modified gravity-stamps.*
(2) *Steam-stamps.*
(3) *Arrastras and pans.*
(4) *Centrifugal and edge-runner mills.*
(5) *Tube-mills and ball-mills.*

(1) MODIFIED GRAVITY-STAMPS

From time to time attempts have been made to assist and increase the speed of the stamp-drop with the aid of coiled springs, air-chambers, or vacuum-chambers in connection with the stems. The main object has, of course, been to enable a greater number of blows per minute to be struck than is possible with the gravity stamp. The use of coiled springs has not been successful for mechanical reasons, the springs soon wearing out and losing their resilience. Acceleration with compressed air seems to promise better results and work along this line in South Africa is being carried on vigorously. The Morison stamp, which is driven by cranks in place of cams and is equipped with a hydro-pneumatic accelerating rig, was operated for several months in the Meyer and Charlton mill, S. A. With 1600-pound stamps dropping 135 times per minute a duty of over 9 tons per day was maintained.* The Holman pneumatic-stamp is also of this type. Another air-accelerated stamp weighing 1670 pounds, invented by H. C. Behr on the Rand, is said to have given 140 drops per minute and to be mechanically efficient.†

(2) STEAM-STAMPS

Michigan Steam-Stamps. It is not often that we find the use of a highly successful machine confined to one particular district, but the immense, single, steam-stamp, used almost to the exclus-

* H. S. Denny, Eng. & Min. Jour., Vol. 88, pages 1157–58.
† C. O. Schmitt, Min. & Min., Vol. 30, page 630.

ion of other crushing machinery in the Michigan copper country, is seen nowhere else. These machines, one of which is shown in Fig. 47, consists of a mortar and stamp, the stem of the latter ter-

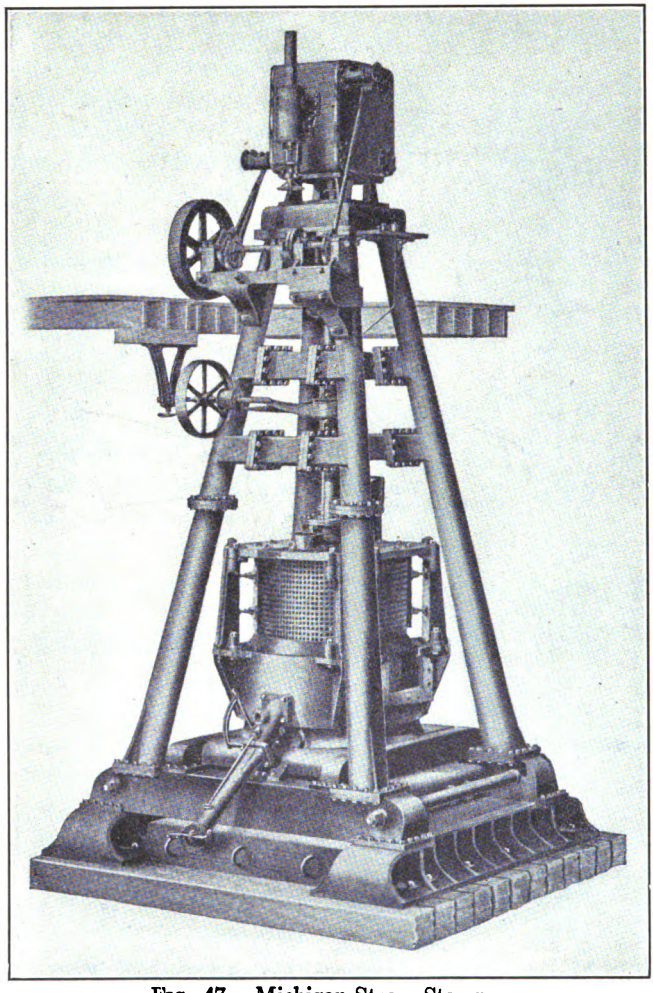


FIG. 47. Michigan Steam-Stamp.

minating in a steam cylinder which serves to raise and lower the stamp. Richards,* comparing the California gravity stamp with the Michigan steam-stamp, gives the energy in foot-pounds of each

* Ore Dressing, Vol. 1, page 132.

as 439 and 32,913 respectively. Each 'head,' as a stamp is called, crushes from 350 to 400 tons in 24 hours through a $\frac{3}{16}$-in. screen. Steam-stamps were tried for crushing and amalgamating gold ores in the Black Hills, S. Dakota,* but for various reasons, many of which could have been remedied, they were a failure from the amalgamating viewpoint. Inside amalgamation was not possible and facilities for outside amalgamation were not properly provided. Where coarse crushing is desired, without regard to amalgamation, the use of steam-stamps would give a compact plant. The power consumed per ton of ore crushed is practically equal in steam and gravity mills.

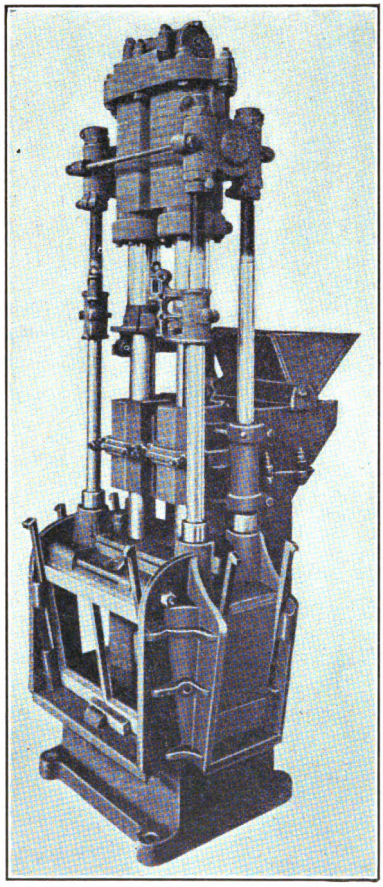


FIG. 48. Tremain Steam-stamp Mill.

Small steam-stamps equivalent in size to the ordinary gravity stamp have been on the market for a number of years, but have not come into general use.

The Tremain Steam-Stamp, shown in Fig. 48, is the best known and most used. It consists of a two-stamp battery with the stamps actuated by steam cylinders. The mortar is fitted with screens on the front and at each end. Inside amalgamation is provided for, but does not appear to have been used with much success. The number of drops ranges from 140 to 200 per minute according to the steam pressure. E. W. Sperry † found the mill a good amalgamator, with a capacity of from 10 to 19 tons per day for the two stamps, using about 12

* T. A. Rickard, Stamp Milling, page 97.

† Trans. A.I.M.E., Vol. 26, page 545.

h-p. Repair costs were low. In Rhodesia, S. A., these mills are popular on account of their portability * and they appear to be easily kept in repair. The fuel consumption is high, but in some localities this is of little consequence.

(3) ARRASTRAS AND PANS

The Arrastra. The forerunner of the grinding-pan was the arrasta (Fig. 49). This is probably the oldest milling device in America. When intelligently operated it is at least as efficient as

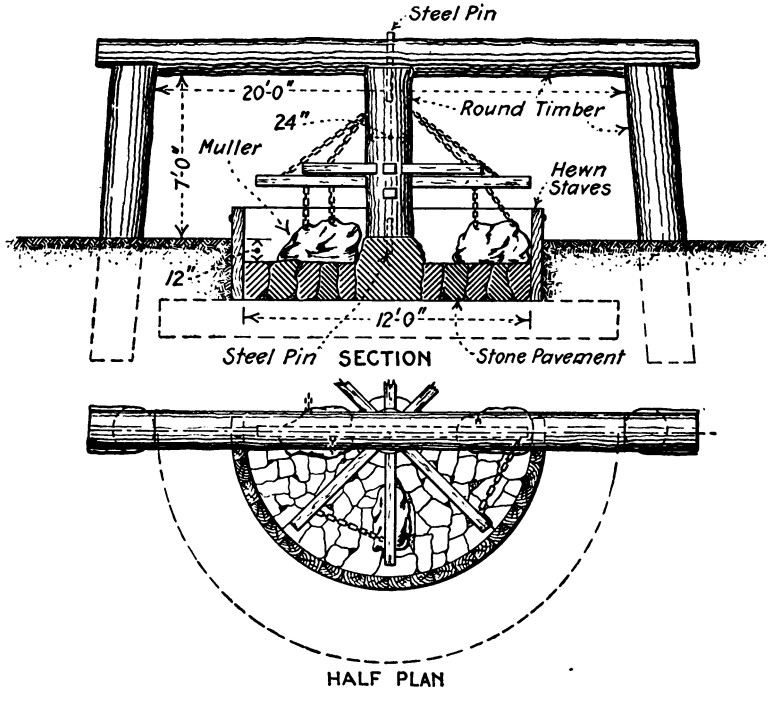


FIG. 49. Arrastra.

a first-class stamp mill, so far as amalgamation is concerned. It is still valuable for prospecting and for the working of small deposits of rich ore. The arrastra consists of a circular enclosure paved with stones; upon this pavement, heavy mullers, also of stone, are dragged by means of arms extended from the central spindle. The spindle is revolved either by animal or mechanical

* C. E. Parsons, Min. & Sci. Press, Vol. 97, pages 386–89.

power. The pavement is composed of flat hard rocks about a foot thick, preferably set in cement, but as cement is seldom available in regions where arrastras are used, clay is generally substituted. A rim or wall of rock or wooden pickets is built around the outside of the ring, which is usually from 10 to 20 ft. in diameter. The mullers, from 2 to 8 in number, are as heavy as can conveniently be handled. They often weigh about 600 to 700 lb. and are set with their forward edges slightly raised in order to ride over the coarse grains in the ore. The charge of ore, usually from 1000 to 3000 lb., is placed in the arrastra-bed and is ground with a small amount of water for several hours. Mercury is then sprinkled over the entire charge. The exact dilution of pulp at this point is all-important as it must be sufficiently thin to work readily and yet thick enough to keep the mercury in suspension. The mill is again started, and is run until no more free gold is seen on panning. The pulp is then diluted and the muller is revolved rapidly to keep in suspension the worthless gangue matter, which is run out through plug-holes in the side, into a sluice-box, or over copper-plates to catch escaping amalgam. The great bulk of the amalgam is left behind with the concentrate on the bed of the arrastra from which it is gathered up and cleaned by panning.

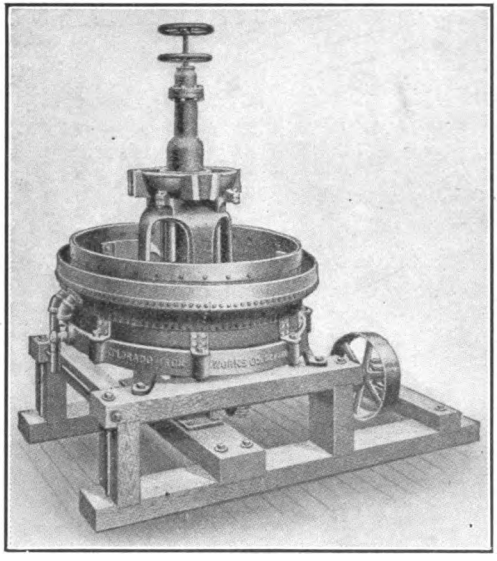


FIG. 50. Grinding Pan.

The Grinding-Pan is the modern successor of the arrastra, in which the stone pavement is replaced by a cast-iron pan, and the heavy stone mullers by shoes which gain their grinding action not by weight but by pressure of a screw feed (Fig. 50). These pans are much less used than formerly, when they were an important feature in the amal-

gamating of silver ores. The grinding action was often omitted in this case, the mullers being used mainly to assist in agitation and amalgamation. In recent years the pan, as a fine-grinder preliminary to cyanide treatment, has been used extensively in Australia and good results are reported. In the United States and Mexico, however, it has met with less favor than the tube-mill.

The figure shows the Australian type of pan known as the Wheeler, the drawing being that of a mill furnished to the Homestake Company of S. Dakota by the Colorado Iron Works.

(4) CENTRIFUGAL AND EDGE-RUNNER MILLS

The Huntington Mill is perhaps the strongest competitor the stamp mill ever had in its field of combined crushing and amalgam-

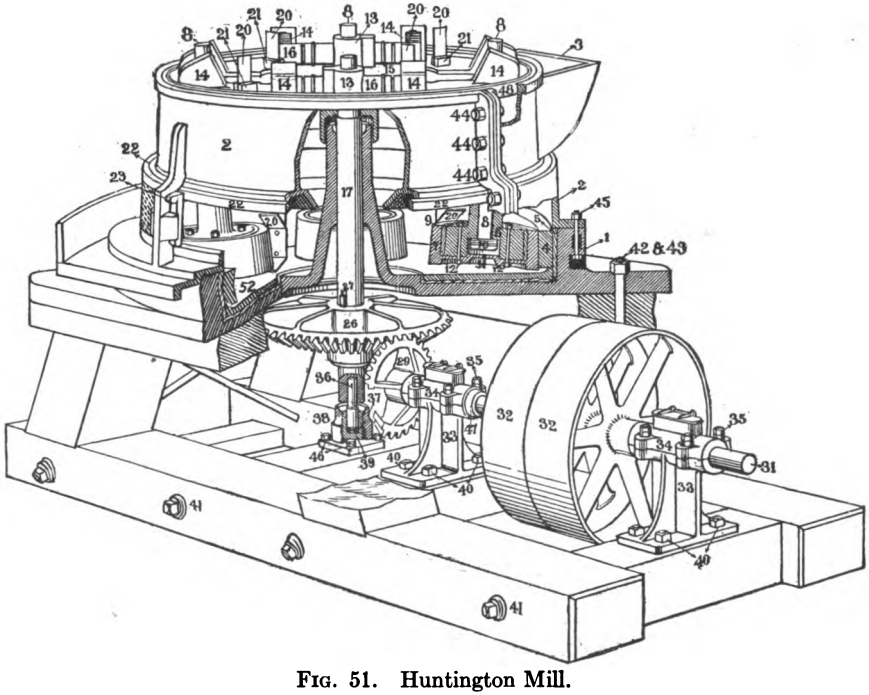


FIG. 51. Huntington Mill.

ating. In this mill (Fig. 51) the crushing is done by the centrifugal force of steel rollers or mullers operating on the inside of a steel die-ring. These rollers or mullers are hung on a spider and are free to revolve on their own axes as well as to rotate about the

main, central spindle. The die-ring is keyed with wooden wedges inside a cast-iron pan. The pan is surmounted by a light cast-iron housing with suitable screen and feed openings. In the center of the pan-bottom is a conical pillar for the reception and guiding of the spindle, which is rotated from below by a bevel-gear and belt-drive. As the spider, yoke, and mullers revolve, the latter are thrown out pendulum-like against the die-ring and crush the ore both by pressure and by impact. To aid the pressure of the mullers against the die-ring, the muller spindles incline inward and the faces of the mullers are correspondingly beveled. The mill is made in several sizes, particulars of which are given in the accompanying table:

TABLE III

Size of mill, nominal diameter	Diam. of die ring	Diam. of muller	Wt. of mill	Capacity on ½-in. material	Power required	Speed
ft.	ft. in.	in.	lb.	tons	h. p.	r.p.m.
3½	3 4	14½	7,000	5 to 10	5 to 10	90
5	4 9	16½	15,000	10 to 20	10 to 12	70
6	5 5½	19	25,500	20 to 40	12 to 20	55
6 (heavy)	5 5½	19	44,600	40 to 80	15 to 30	55

When used as an amalgamator a large quantity (25 to 50 lb.) of mercury is charged into the mill, which is usually provided with a quicksilver-outlet in the pan-bottom. The Huntington is a good amalgamator as long as there are no derangements within the mill, but loosening of any of the parts, coarse pieces of rock, overfeeding or other irregularities are likely to result in the amalgam being swept through the screen. For crushing soft, clayey ores it is superior to the stamp. If hard ore is fed, it must be crushed to at least ½ in. diameter. The Huntington mill is efficient in its own field when intelligently looked after, but it requires much closer supervision than the stamp mill and is by no means foolproof. For this reason it is losing popular favor. Huntingtons should be stopped a few minutes at the end of every shift, the interior of the mill rapidly but thoroughly inspected and bolts, mullers, and die-rings sounded with a hammer to detect looseness. If anything is wrong with one of the mullers, do not attempt to repair it in place, but lift it out with the chain-block and put a spare one in its place. In this way little time is lost, and the defective muller can be repaired at leisure and to much better advantage than in the interior of the mill. In the case of a large plant

where a complete shutdown is out of the question an extra mill is provided so that the repair gang can go over the entire series of Huntingtons every day.

The upkeep of these mills is higher than that of stamps; screen repairs are excessive and only punched-metal screens are suitable. In first cost Huntingtons are cheaper than stamps; they are also more readily erected and more easily moved from place to place.

Edge-runner or Chilean Mills have greatly increased in popularity in recent years. In primitive times crude mills of this type were used to prepare the ore for the arrastra. At present they are used in some cases to replace stamps, but usually follow the stamps as regrinding machines.

In principle, the 'Chile' mill, as it is popularly called, consists of two or more large steel rollers rotating about a central spindle and

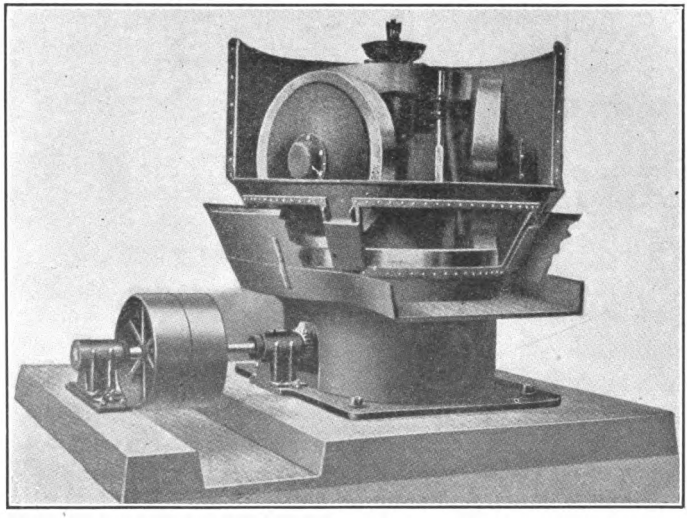


FIG. 52. High-speed Chilean Mill.

rolling on a circular steel track. The discharge is by centrifugal force through peripheral screens similar to those in the Huntington mill. The middle of the tires does pure crushing; the edges do both crushing and grinding. Chilean mills may be separated into the high-speed and low-speed types.

A typical high-speed mill (Fig. 52) will have from three to four rollers, 4 to 6 ft. in diameter by 8 in. face, making 25 to 35 rev. per min. The die or track will be from 5 to 7 ft. internal dia-

meter. The capacity of such a mill is from 25 to 75 tons per day, using 25 to 40 h. p. High-speed mills are used for regrinding in concentrating mills and to follow the stamps in some cyanide plants.* They are unsuited for inside amalgamation but may be advantageously followed by plate amalgamation.

The best-known slow-speed machine of this type is the Lane Chilean mill (Fig. 53). C. C. Lane gives the following data of a successful mill of this kind in operation at Barron, Washington: Six runners, diameter 42 in., width at center 9 in., at edge 5 in. Weight of each runner or wheel 900 lb. Diameter of track outside 10 ft., speed 6 to 8 rev. per min. Size of feed $\frac{1}{4}$ in., size of

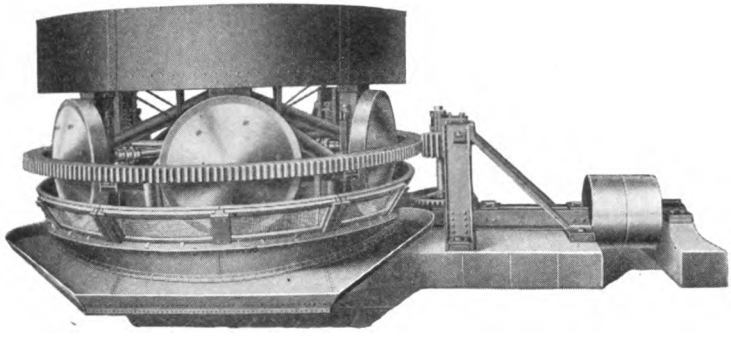


FIG. 53. Lane Slow-speed Chilean Mill.

product 70 mesh, average capacity 40 tons per 24 hours, using 12 h. p. The pan above the wheels is loaded with stones or scrap-iron to give weight to assist the grinding. These slow-speed mills have given satisfaction as inside amalgamators, the action in this respect being not unlike that of the arrastra. The discharge in the slow-speed mill is by overflow through a coarse (10-mesh) screen, the size of product being regulated by the height of discharge and not by a change of screens.

(5) TUBE-MILLS AND BALL-MILLS

The Tube-Mill, long a popular machine in the cement industry, has, in recent years, become an important factor in the fine-grinding of ore preliminary to cyanide treatment.

Mechanically, the tube-mill is extremely simple. It consists of a sheet-steel cylinder, from 3 to 6 ft. in diameter by 10 to 20 ft.

* J. W. Hutchinson, Min. & Sci. Press, Vol. 102, pages 617 and 619.

long, set with its axis at a slight angle with the horizontal. This cylinder is given a suitable lining and after being half filled with hard pebbles, from $\frac{3}{4}$ to 4 in. in diameter, is slowly revolved by suitable mechanism. Thick pulp (when, as is usually the case, wet crushing is employed) is fed into the upper end of the mill and

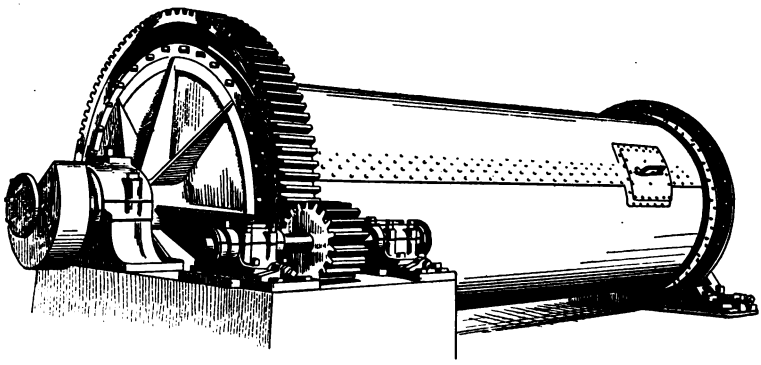


FIG. 54. Trunnion-type Tube-Mill with Scoop Feed.

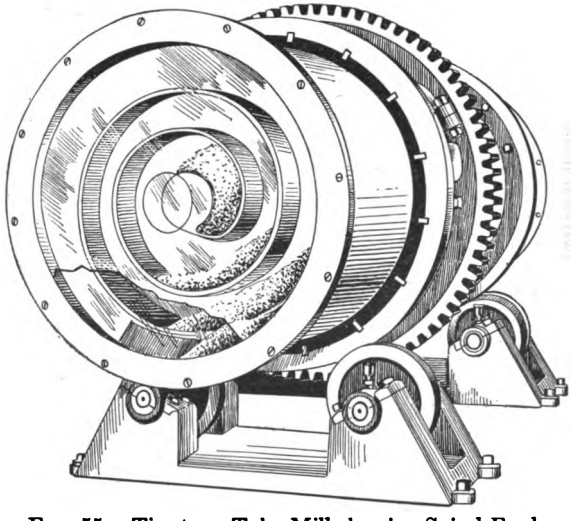
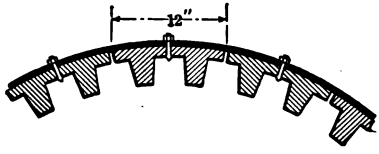


FIG. 55. Tire-type Tube-Mill showing Spiral Feed.

after being reduced to a fine state of division by the action of the pebbles is discharged at the lower end. The feeding is accomplished either through a scoop communicating with a hollow trunnion (Fig. 54) or by a spiral feed on the principle of an Archimedean screw, see Fig. 55.

The proportion of water to solid is an important factor in tube-mill efficiency, and, up to a certain point, as the water in the pulp decreases the capacity of the mill increases rapidly. This point of 'critical dilution,' as it has been called, apparently varies slightly with different ores, but a pulp with from 35 to 45 per cent moisture seems to give the best results. To ensure the proper consistency, tube-mills are almost always fed from a classifier, which serves also to remove the material that is already crushed to the proper fineness.



Enlarged Section

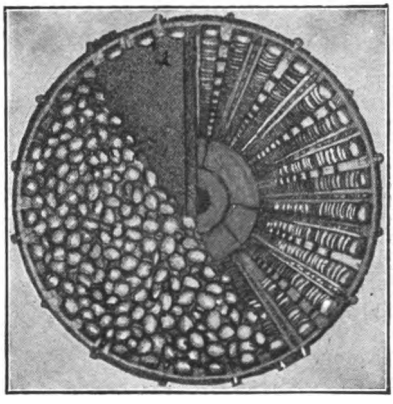


Fig. 56. El Oro Tube-mill Lining.

The discharge is usually by means of a hollow trunnion, although in some mills a spiral similar to the feeding spiral is used. In this event, slots in the end of the mill communicating with the spiral allow the pulp to flow out while the pebbles are retained. The trunnion discharge has the effect of a classifier and heavy mineral grains are necessarily crushed finer than the grains of gangue before they are discharged. This would usually be a distinct advantage where the pulp is to be cyanided. No attempt is made to screen the tube-mill product; instead it is generally sent to a classifier and the sand product returned to the mill for regrinding.

For lining, cast-iron, cast-steel, and flint blocks have been used; preference is given to the lining of flint or 'silex,' as it is called. The pebbles are frequently a variety of flint, found in Greenland, though where hard ores are found locally, their use would seem to be desirable for this purpose. An extremely ingenious device, known as the El Oro lining, has recently been placed on the market. It consists, see Fig. 56, of a series of ribbed plates secured to the inside of the mill; the projecting ribs form tapering channels into which the pebbles are forced and wedged into place by the action of the mill. As the pebbles wear they drop out of the slots and are

Tube-mill 5 ft. diam. half full of pebbles, speed 30 rev. per min. Lining $4\frac{1}{2}$ in. thick, pebbles assumed as 3-in. spheres.

For any pebble centrifugal force $= \frac{mV^2}{r}$ (m = mass, V = velocity, r = radius) and gravitational force when resolved along the radius is opposed to centrifugal force and is equal to $mg \sin \theta$. (The tangential component of gravity, $mg \cos \theta$, is overcome by the work done in revolving the mill and serves to keep the pebbles pressed together.)

When $\frac{mV^2}{r} = mg \sin \theta$,

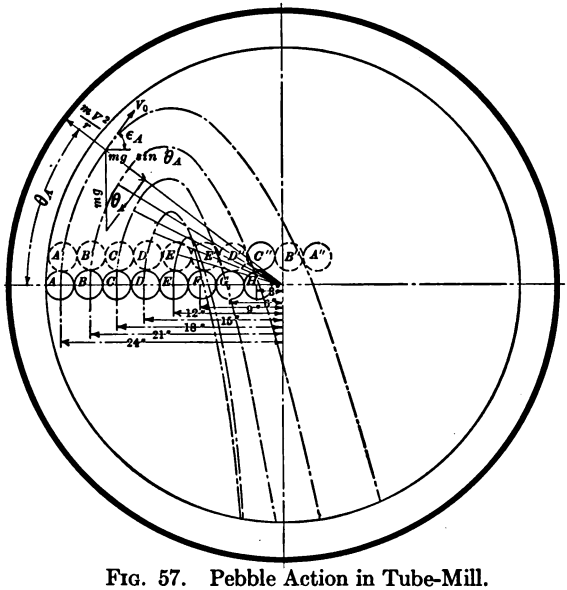
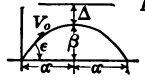


FIG. 57. Pebble Action in Tube-Mill.

the pebble is no longer forced outward but is free to take the tangential push imparted to it by the wall of the mill and by the pebble following it; and is launched into space as a projectile with an initial velocity V_0 equal to its rotational speed, and at an angle $\epsilon = 90 - \theta$.

(1) To find θ. $\frac{mV^2}{r} = mg \sin \theta$, but $V^2 = (2\pi rn)^2$, where r = radius and n = rev. per sec. Therefore $\sin \theta = \frac{4\, r\pi^2 n^2}{g}$ ($g = 32.2$, $\pi^2 = 9.9$, $n = \frac{1}{2}$) and $\sin \theta = 0.308\, r$.

(2) To find the parabola.



$$\alpha = \frac{V_0^2}{2g} \sin 2\epsilon,$$

$$\beta = \frac{V_0^2}{2g} \sin^2 \epsilon$$

$$\Delta = \frac{V_0^2}{2g} \cos^2 \epsilon.$$

For each ball $V_0 = 2\pi rn$, $\epsilon = 90° - \theta$.

PARABOLAS FOR BALLS *A* TO *E*

Ball	Angle θ $\sin \theta = 0.308\, r$	Angle ϵ $\epsilon = 90° - \theta$	$\alpha = \frac{V_0^2}{2g} \sin 2\epsilon$	$\beta = \frac{V_0^2}{2g} \sin^2 \epsilon$	$\Delta = \frac{V_0^2}{2g} \cos^2 \epsilon$
A	38	52	0.592	0.378	0.231
B	32	58	0.422	0.338	0.132
C	$27\frac{1}{2}$	$62\frac{1}{2}$	0.279	0.267	0.072
D	$22\frac{1}{2}$	$67\frac{1}{2}$	0.163	0.196	0.034
E	18	72°	0.090	0.140	0.015

From an inspection of the figure it is evident that balls *A*, *B* and *C* will not be free to fall in their natural parabolic paths, owing to the crowding of the balls nearer the center. For example the ball *E* will rise to position *E'* and will crowd its predecessor which is on the way down, over to position *E''*; this in turn crowds *D''* and so on, with the result that *A''* occupies a position outside of its normal parabola. If we consider balls *G* and *H*, this crowding will be still more marked. The general result, therefore, is a rolling of the outer pebbles down a slope much flatter than the parabola and a crowding together of the pebbles on the interior of the mass. (99)

automatically replaced. This lining is cheaper and better than its next-best competitor the silex blocks, which are cut key-fashion and are cemented around the interior of the mill.

The action of the tube-mill is somewhat as follows: In the rotation of the mill the pebbles next to the shell, see Fig. 57, cling to the side, the height to which they rise being regulated by the speed of the mill. As long as the centrifugal force is greater than the gravity component, these pebbles will continue to revolve with the walls of the tube-mill until a point is reached — indicated by the angle θ in the figure — where these pebbles leave the wall and fall in a parabolic path, striking and rolling over the other pebbles below and crushing, both by impact and by attrition. Each succeeding layer of pebbles nearer the center of rotation has less centrifugal force and does not rise as high. The pulp fed into the mill is also carried up as a coating on the pebbles, and by the constant rubbing and striking of the pebbles against each other and against the lining is reduced to an extremely fine state of division. From the figure, however, it is evident that impact is a less important factor in the work of the mill than would at first be supposed. If the speed of a tube-mill is increased to a point where the entire mass of pebbles distributes itself evenly about the walls, *i.e.*, where the centrifugal force throughout the mill exceeds the component of gravity, crushing ceases, and the power-consumption of the mill drops to almost nothing. Table IV shows the performance of the tube-mill in various cyanide plants.

The Hardinge * Pebble-Mill differs from the foregoing in that the cylinder is replaced by two cones, placed base to base, with a short cylinder between, see Fig. 58. It is claimed for this mill, since the pebbles range themselves in decreasing size from the maximum mill-diameter to the minimum mill-diameter, that the size of the pebbles, and hence their crushing effect, is in proportion to the size of the particles to be crushed in the various parts of the mill. In other words, the coarse particles are crushed by the large pebbles and the fine particles are crushed by the small pebbles. While the machine has not been tested sufficiently in competition with the cylindrical tube to allow of any decided comparison, it seems to promise well for all but extremely fine grinding.

Tube-mills are not usually considered as amalgamating ma-

* Trans. A.I.M.E., Vol. 39, pages 336–341.

TABLE IV

TUBE-MILL PERFORMANCES

Ref.	Size	H. P.	R.P.M.	Cap. tons	Feed	Product	Remarks
1...	4 ×16 ft.	.....	.....	70	10 mesh (*a*)	(*b*)	Iron liners cost 14¢ per ton ore
2...	5?×16 ft.	30	32?	38	20%–40 mesh	–150 mesh	Iron liners
3...	5?×13 ft.	20	32	19½	51%–40 mesh	–150 mesh	Iron liners
4...	5?×22 ft.	50	27½	77	15%–40 mesh	73%–150 mesh	Total cost per ton 28¢, 4 lb. pebbles per ton. 11,000 lb. pebbles charge
5...	4½×20 ft., A.	43 (*c*)	—	80	65%–100 mesh	51%–120 mesh	¼ lb. pebbles per ton
6...	4½×19½ ft., A.	51	—	96	40%–80 mesh	75% –100 mesh	11,000 lb. pebbles
7...	4 ×19½ ft., K.	48	—	96	32%–80 mesh	96%–100 mesh	11,000 lb. pebbles
8...	6 ×12 ft., C.	15	—	72	63%–20 mesh	91%–100 mesh	2,000 lb. pebbles
9...	5 ×23 ft.	51	—	172	'12 mesh'	55% –150 mesh	19,000 lb. pebbles
10...	5 ×23 ft.	19	—	172	'12 mesh'	22%–150 mesh	3,000 lb. pebbles
11...	5 ×23 ft.	75	—	172	'12 mesh'	57% –150 mesh	25,000 lb. pebbles
12...	4½×20 ft., A.	—	—	65	'26 mesh'	79%–120 mesh	Total cost 25¢
13...	5 ×20 ft., A.	50	—	120	'10 mesh'	'100 mesh'	
14...	4 ×18 ft., K.	33	31	104	30%–100 mesh	78%–150 mesh	El Oro lining
15...	5 ×15 ft., A.	27	29	158	26%–100 mesh	57%–150 mesh	Silex lining
16...	4 ×18 ft., A.	27	31	110	25%–100 mesh	69%–150 mesh	Silex lining
17...	5 ×20 ft.	51	26½	103?	26%–100 mesh	79%–150 mesh	El Oro lining
18...	5 ×24 ft., A. C.	—	26	120	'16 mesh'	49%–150 mesh	
19...	4½×20 ft., A.	25	25	75	58% 100 mesh	64%–120 mesh	16,000 lb. pebbles
20...	8 ft., Con.	30	30	75	79% 100 mesh	32%–120 mesh	7,000 lb. pebbles
21...	5×22 ft., A. C.	59½	—	100	74% 100 mesh	51%–200 mesh	Silex lining — consumption 0.6 lb. per ton, cost 2.3¢. Danish pebbles 1.8 lb. per ton (*d*)
22...	5 ×22 ft., A.	.....	.....	125	'10 mesh'	'100 mesh'	Cost 9 to 20¢ per ton of ore
23...	6 ×8 ft.	.....	.....	24	'8 mesh'	'125 mesh'	
24...	5½×22 ft.	60	.....	150	.....	.....	10,000 lb. pebbles, 6-in. Silex lining

Reference		Reference	
1...	L. D. McMiken, Min. & Sci. Press, Vol. 93, page 534.	13...	E. B. Wilson, Min. & Min., Vol. 29, page 8.
2...	H. T. Brett, Min. & Sci. Press, Vol. 93, page 744.	14...	V. B. Sherrod, Min. & Sci., Press, Vol. 100, page 367.
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4...	E. G. Banks, Trans. A.I.M.E., Vol. 38, page 196.	16...	V. B. Sherrod, Min. & Sci. Press, Vol. 100, page 367.
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6...	H. W. Hardinge, Inventor of Conical Mill, Min. Wld., Vol. 27, page 1005.	18...	Walter Neil, Min. & Sci. Press, Vol. 100, page 483.
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8...	H. W. Hardinge, Inventor of Conical Mill,	20...	Stuart Tod, Min. & Sci. Press, Vol. 101, page 243.
9...	H. W. Fox, Min. & Min., Vol. 28, page 537.	21...	J. W. Hutchinson, Min. & Sci. Press, Vol. 102, page 620.
10...	H. W. Fox, Min. & Min., Vol. 28, page 537.	22...	R. H. Richards, Ore Dressing, Vol. 3, page 1291.
11...	H. W. Fox, Min. & Min., Vol. 28, page 537.	23...	R. H. Richards, Ore Dressing, Vol. 3, page 1291.
12...	E. B. Wilson, Min. & Min., Vol. 29, page 8.	24...	R. H. Richards, Ore Dressing, Vol. 3, page 1291.

(*a*) On battery. (*b*) To equal 35 mesh on battery. (*c*) 60 H. P. to start.

(*d*) Cost: Labor..... 1.4¢ per ton.
Supplies..... 6.5 per ton.
Power..... 8.7 per ton.
Total..... 16.6 per ton.

A., Abbé. K., Krupp. C., Conical. A. C., Allis-Chalmers. Con., Hardinge Conical.

chines, but recently several metallurgists * have claimed an increased extraction, due to feeding quicksilver into their tube-mills.

Ball-Mills are in principle similar to, but not identical with, tube-mills. Forged-steel balls and manganese-steel plates are

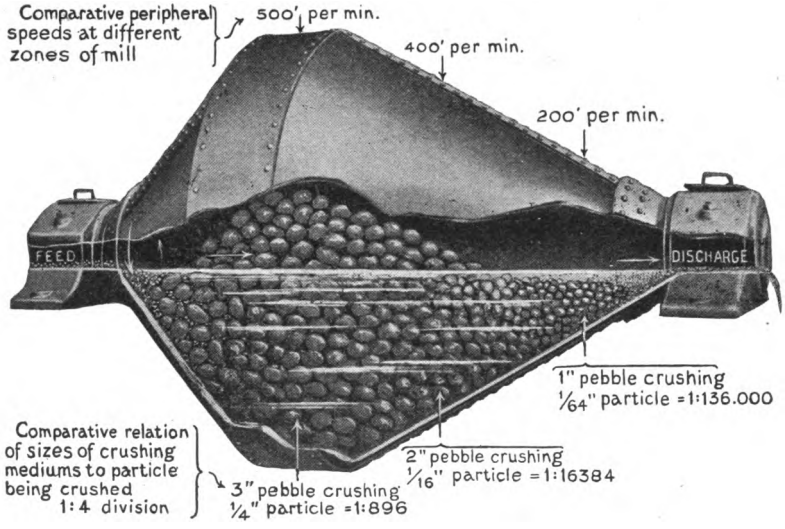

FIG. 58. Hardinge Conical Pebble-Mill.

substituted for pebbles and silex linings. The action within the mill is also to some extent different, in that the balls are comparatively large, few in number, and do not rise as high as the pebbles in the tube-mill. The ball-mills are usually larger in diameter and shorter than the tube-mills and do not revolve as rapidly. They are generally used for dry-crushing. Discharge is provided for by slots between the overlapping edges of the lining-plates. These slots communicate with an annular space around the mill, which is covered with two layers of screen, a coarse screen to take the heavy wear, and a fine screen to limit the mesh of the product, see Fig. 59. The oversize is returned to the mill in the course of its revolution. Ball-mills have never been popular in the United States, but in Australia they have done good service as grinders following coarse rock-breakers.

The advantages of the ball-mill, as summarized by Eissler † in 1895, still seem to hold good. They are:

* Min. & Sci. Press, Vol. 100, page 608.

† The Metallurgy of Gold, 4th Ed., page 148a.

(1) "Comparatively large-sized pieces of ore, the size of a double-fist, can be fed into it."

(2) "It requires little or no supervision and very small horse-power."

(3) "The wear and tear of the mill . . . is relatively very small for the work done by it."

(4) "The internal parts of the mill are always easily got at."

(5) "It crushes very regularly to any mesh of sieve required and does not make too many slimes."

(The last statement is less cogent to-day than in 1895.)

To this list might be added, that the moment the ore is crushed

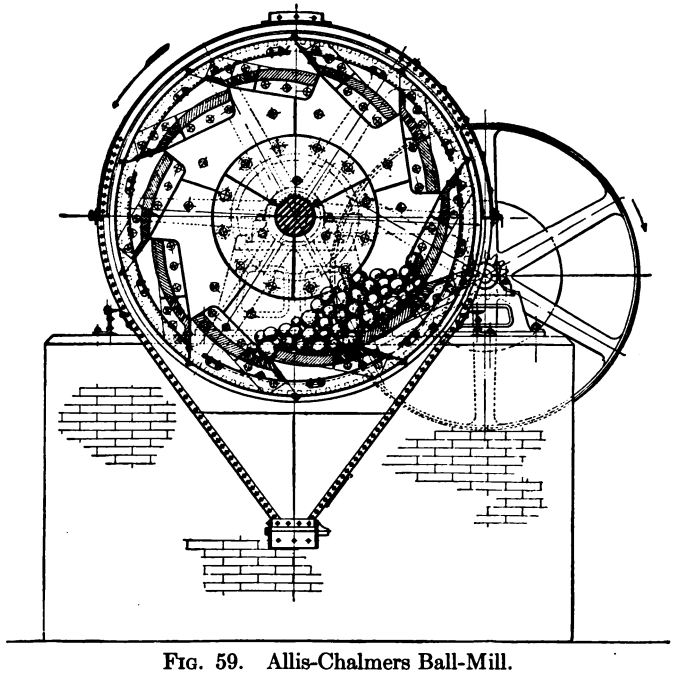

FIG. 59. Allis-Chalmers Ball-Mill.

fine enough, it must pass out of the mill owing to the efficiency and large area of the screen-surface.

It is strange that a machine apparently so efficient should not be in more general use, but American manufacturers are beginning to invade the ball-mill field, which heretofore has been almost entirely in the hands of the German iron-works, and several prominent metallurgists are projecting plants involving the employment of these evidently efficient comminutors.

The data for the following table are taken from Australian practice, mainly as outlined in an excellent article on 'Ball-mill Practice' by M. W. von Bernewitz.*

TABLE V

Size of mill	Weight of balls	H. P. required	Capacity tons per 24 hours	Ball consumption, lb. per ton	Total cost, ¢ per ton	Character of feed	Character of product
No. 5	....	24	33	0.5	..	Fairly soft, calcite and quartz 75%, 2-in. ring..	24%+ 40 mesh 46%−200 mesh 43%−150 mesh
No. 5	2350	18	40	0.32	..	"Hard schistose 3-in. feed"	
No. 5	2800	26	....	0.47	48	"Hard 60% SiO_2, 3-in. feed."	55%−150 mesh
No. 8	4400	52	100	0.65	53	Schistose, 3-in. feed.	54%−150 mesh
No. 8	4480	57	95–100	0.47	64	Schistose, 3-in. feed.	55%−150 mesh
No. 8	4480	52	80–90	0.64	35	"Very hard cherty, 3-in. feed."	1%+ 30 mesh 52%−150 mesh
Wet crushing							
No. 5	2200	25	30	....	..	"Hard dense quartz"	23%+ 30 mesh 25%−150 mesh
No. 5	2400	..	93–100	(1)	..	.....	10%+ 40 mesh 58%−150 mesh

All mills are of Krupp manufacture.
No. 5 is 89 by 46 in., is driven at 25 rev. per min., and has 70 sq. ft. screen area.
No. 8 is 106 by 54 in., is driven at 24 rev. per min., and has 100 sq. ft. screen area.
(1) Double the wear when dry-crushing.

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CHAPTER VII

COMPARISONS OF VARIOUS MILLS

A discussion of this kind would be somewhat pointless without some comparison of the various crushing devices mentioned. It is obvious that the most efficient crushing machine is that which produces the desired result with the least expenditure of energy for operation, and of time and money for repairs and renewals. Furthermore, comparisons can only be made between pieces of apparatus that cover in some part at least the same or similar territory. It is scarcely reasonable to compare the work of a gyratory crusher, reducing mine-run ore to 3-in. size, with that of a tube-mill, reducing 20-mesh material to 200-mesh. Between the machines discussed in the last two chapters, however, comparison is legitimate and, as has been said, necessary.

Three points of relative merit may be established for purposes of discussion:

(1) *Reduction in size effected per unit of power.*
(2) *Suitability of product for purposes in hand.*
(3) *Wear and tear on various parts, and loss of time incident thereto.*

(1) **Reduction in Size Effected per Unit of Power** — this is commonly referred to as 'crushing efficiency.' An adequate discussion of this extremely important question cannot be here attempted.

Certain work done during the last few years in connection with the efficiency of pre-cyanide crushing * has placed in our hands a simple and apparently logical basis for rough comparisons.

Stadtler's hypothesis, briefly stated, is, that in crushing an ore, the area of fracture, multiplied by a coefficient determining the resistance, represents the *force* required to cause the fracture.

* R. W. Chapman, Proc. Aust. Inst. Min. Eng., Oct., 1909; abs. Min. Mag., Vol. 2, page 64. H. Stadtler 'Grading Analyses,' Inst. Min. & Met., May, 1910; J. C. M. & M. Soc. S. A., May, 1910; abs. Min. & Min., Vol. 31, page 344.

In order to perform mechanical work this *force* has to move through a *distance* represented by the deformation which the body can stand before reaching the breaking point. It is in this connection, immaterial that this distance of deformation — within the limits of elasticity and plasticity — is, for non-homogeneous bodies, subject to variations, because for highly elastic and unplastic bodies, as quartz, glass, etc., these variations are too insignificant to be considered. In addition they are by the nature of our crushing appliances averaged to such an extent, that these averages are as good as exactly defined figures, and dealing with relative values only, we do not need to know the exact extent of this deformation. It is sufficient for our purpose that this factor is a constant function of the diameter of the particles to be crushed.

The mechanical work done is represented by the product of the force by the distance; but, as in a regular scale of reduction by volume, in which $r = \frac{1}{2}$, the diameters of the particles decrease at the same ratio as the area of fracture increases; the product, or the mechanical work required for reducing the volume (or weight) of the unit from one grade to the next following, is a constant for each grade, called the crushing or energy unit (E. U.).

The same conclusion is also arrived at by the application of Kick's Law, which reads:

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state, varies as the volumes or weights of these bodies."

The volumes of the particles decrease from grade to grade, at the same ratio as the number of the particles increases, said particles constituting in their total the volume of the unity; and the product of the volume into the number of the particles of that grade is, therefore, constant for each grade. Since, in conformity to the above law, the amount of energy absorbed is proportional to the volume of the body to be crushed, it follows again also that the total energy required for reducing the weight of the unit is constant for each grade.

The ordinal numbers of any arithmetical progression given to these grades represent consequently the relative values of the energy which has to be spent upon producing the respective grade from the initial unit; *i.e.*, the mechanical value of the grade in energy units (E. U.).

For obtaining the mechanical value of mixed sands we need only to multiply the percentages of the gradings by the number of the energy units of the respective grade and add the products. This possibility of having the grading of pulps condensed and expressed in one representative figure proves to be of great value.

The useful work done per unit by any crushing machine is determined by the difference between the mechanical values of the samples taken at the inlet, and at the discharge of the machine, and for obtaining the total work done this difference has to be multiplied by the tonnage handled.

The relative mechanical efficiency is the value obtained by dividing the total work done by the unit of energy (for instance horse-power):

Relative mechanical efficiency

$$= \frac{\text{Tonnage} \times \text{work done per unit in E. U.}}{\text{Unit of energy (h. p.).}}$$

The appended table shows the data compiled by Mr. Stadtler for his work and is self-explanatory. The right-hand side of the table is of value in other connections also, in giving the data of the now generally adopted 'I.M.M. screens.'

TABLE VI

Standard Grades and Screens

Mines Trials Committee—(M.T.C.) standard grades								Institution of Mining and Metallurgy. Standard laboratory screens					
Ordinal number or mech. value	Mesh aperture (=sides of theoret. cubes)		Volume (or weight) of theoret. cubes		Area of fracture from unit	Commercial denomination of screens — Approx. number of meshes		Ordin. number or mech. value of grade		No. of mesh	Mesh aperture	Dia. of wire	Area disch'ge
E.U.	in.	mm.	$\frac{1}{1000}$ cu. in.	cu. mm.	sq. in.	p. lin. in.	p. sq. in.	E.U.		lin. in.	in.	in.	%
30	.00098	.02480	.000000	.00002	3072.0			30.0					
29	.00123	.03125	.000001	.00003	2438.1	–200 grade incl. washed slimes		29.0					
28	.00155	.03937	.000003	.00006	1935.4			28.0					
27	.00195	.04961	.000007	.00012	1536.0			27.0					
26	.00246	.06250	.000015	.00024	1219.1	200	40000	26.0	26.02	**200**	.0025	.0025	25.0
25	.00310	.07875	.000030	.00049	967.68	150	22500	25.0					
24	.00391	.09922	.000059	.00098	768.0	130	17000	24.0	24.81	**150**	.0033	.0033	24.50
									23.77	**120**	.0042	.0041	25.40
23	.00492	.1250	.000119	.00195	609.52	110	12000	23.0	23.01	**100**	.005	.005	25.0
									22.60	**90**	.0055	.0055	24.50
22	.00620	.15750	.000238	.00391	483.84	**90**	8000	22.0	22.08	**80**	.0062	.0063	24.60
									21.49	**70**	.0071	.0071	24.70
21	.00781	.19844	.000477	.00781	384.0	75	5500	21.0	20.81	**60**	.0083	.0083	24.80
20	.00984	.250	.000954	.01562	304.76	**60**	3500	20.0	20.0	**50**	.01	.01	25.0
19	.01240	.3150	.001907	.03125	241.92	50	2500	19.0	19.03	**40**	.0125	.0125	25.0
18	.01562	.39688	.003815	.06250	192.0	40	2000	18.0					
									17.80	**30**	.0166	.0167	24.8
17	.01969	.50	.007629	.1250	152.38	35	1200	17.0	16.99	**25**	.020	.020	25.0
16	.0248	.630	.015259	.250	120.96	30	900	16.0	16.02	**20**	.025	.025	25.0
15	.03125	.7938	.030517	.50	96.0	25	600	15.0	15.06	**16**	.0312	.0312	24.92
14	.03937	1.0	.061035	1.0	76.19	20	400	14.0					
									13.81	**12**	.0416	.0417	24.92
13	.04961	1.260	.122070	2.0001	60.480	18	300	13.0	13.01	**10**	.05	.05	25.0
12	.06250	1.5875	.244140	4.0002	48.0	14	200	12.0	12.08	**8**	.062	.063	24.60
11	.07875	2.0	.488281	8.0003	38.102	11	120	11.0					
10	.09922	2.520	.976562	16.000	30.240	8	65	10.0	10.0	**5**	.10	.10	25.0
9	.1250	3.1750	1.9530	32.001	24.0	6	35	9.0					
8	.15750	4.0003	3.9060	64.003	19.051	4	15	8.0					
7	.19850	5.040	7.8120	128.02	15.120	3.5	12	7.0					
6	.250	6.350	15.625	256.03	12.0	Ap'rt're $\frac{1}{4}''$	9	6.0					
5	.3150	8.0005	31.250	512.06	9.526	$\frac{5}{16}''$	6	5.0					
4	.39690	10.080	62.50	1024.1	7.560	$\frac{3}{8}''$	4	4.0					
3	.50	12.70	125.0	2048.3	6.0	$\frac{1}{2}''$	3	3.0					
2	.630	16.0	250.0	4096.5	4.763	$\frac{5}{8}''$	2	2.0					
1	.79370	20.160	500.0	8193.0	3.780	$\frac{1}{4}''$	1	1.0					
0	1.00	25.4	1000.0	16386.0	3.0	1 inch.	—	0.0					

Arithmetical progression $R=1$

Geometrical progression.

$R = \frac{1}{\sqrt[3]{2}} = 0.7937$ | $R = \frac{1}{2} = 0.5$ | $R = \sqrt[3]{2} = 1.2599$

The M.T.C. grades are based on the assumption of a successive reduction in volume (or weight) of the screen products by one-half from grade to grade.

As examples of the application of this crushing efficiency calculation we have:

(1) A stamp mill, with 1150-lb. stamps consumes 2.6 h-p. per stamp with a duty of 9 tons per 24 hours. Battery screen 8-mesh, aperture 0.097 in.

TABLE VII

		Feed	Discharge pulp	
Denomination of screen	Mechanical value of mean grade		Grading analyses	Mechanical value of discharge
Number	E. U.		Per cent	E. U.
+ 12	13	Subject to variations, but	19.9	2.59
+ 20	15	by practical experience it	12.1	1.82
+ 30	17	has been ascertained that	9.7	1.65
+ 50	19	the 0 grade (1-inch screen)	11.5	2.19
+ 80	21	represents a fair average	10.9	2.29
+120	23	size.	2.6	.50
+200	25		4.2	1.05
−200	28		29.1	8.15
			100%	20.34
		Less Mech. value of feed		0.0
		Work done by stamp per unit (say 1 ton)		20.34

Total work done per stamp in 24 hours = 9 tons × 20.34 E.U. = 183.06 E.U.

Relative mechanical efficiency per h-p. = $\frac{9 \text{ tons} \times 20.34 \text{ E.U.}}{2.6 \text{ h-p.}}$ = 70.40%

(2) A 22 by 5½-ft. tube-mill, 30 rev. per min., 100 h-p. Feed, 290 tons pulp + 6.75 tons coarse ore of 30 E. U. value when crushed.

TABLE VIII

		Intake		Discharge	
Denomination of screen	Mechanical value of mean grade	Grading analyses	Mechanical value of intake	Grading analyses	Mechanical value of discharge
No.	E. U.	Per cent	E. U.	Per cent	E. U.
+ 12	13	27.6	3.59		
+ 20	15	15.7	2.36		
+ 30	17	13.7	2.33	7.4	1.26
+ 50	19	19.9	3.78	22.5	4.28
+ 80	21	15.5	3.26	29.0	6.09
+120	23	2.0	.46	6.0	1.38
+200	25	1.6	.40	7.7	1.93
−200	28	4.0	1.12	27.4	7.67
		100%	17.30	100%	22.61
		Less mechanical value of intake			17.30
		Work done per unit, say 1 ton			5.31

Total Work Done by Tube-Mill in 24 Hours	*E.U.*
Reduction of pulp feed = 290 tons × 5.31 E.U. = ...	1539.2
Reduction of banket ore = 6.75 tons × 30 E.U. = ...	202.5
Total work done in 24 hours	1742.4

Relative mechanical efficiency per h-p. $= \frac{1742.4 \text{ E.U.}}{100 \text{ h-p.}} =$ 17.42%

These results, while cited for the purpose of illustrating the method of calculating, give also a striking comparison in this instance at least in favor of the mechanical efficiency of the stamp as compared with the tube-mill, the ratio being 70.4 per cent for the stamp and only 17.4 per cent for the tube.

It is evident that comparative efficiencies can only be obtained by this method upon identical ores, but it is quite generally true that after we get down to the finer sizes, there is less difference in the grinding or crushing characteristics of ores than one would expect.

The following table of comparisons then, while subject to criticism from lack of uniformity of conditions, is at least suggestive:

TABLE IX

Comparative Crushing Efficiencies of Various Mills

Type of mill	Screen analysis										Cap. per unit	H. P. per unit	Total or net work done E.U.	Efficiency
	+10	+20	+40	+60	+80	+100	+120	+150	+200	−200				
(I) *Gravity Stamps:*														
1. Gilpin Co., Colo., 600-lb. stamps, 50-mesh screen	....	....	....	....	....	30	.....	35	35	.....	1	1.02	23.7	23.7%
2. Marysville, Mont., 800-lb. stamps, 30-mesh screen	....	....	....	9	12	10	.....	68		.....	1¼	1.6	23.3	25.2%
3. Telluride, Colo., 850-lb. stamps, 14-mesh screen	....	....	11	29	....	.....	15	.....	15	30	4	2	22.9	45.8%
4. Waihi, New Zealand, 1000-lb. stamps, 40-mesh screen	....	....	....	9	16	4	9	7	55		2.9	2.5	23.5	27.3%
5. Waihi, New Zealand, 1000-lb. stamps, 20-mesh screen	....	....	15	15	....	14	.....	12	43		4	2.5	22.8	36.5%
6. Goldfield Cons., 1050-lb. stamps, 4-mesh screen	15	34	10		....	13	.....	.....	7	20	8	2.5	17.8	60.5%
7. ? S. Africa, 1150-lb. stamps, 8-mesh screen	20	12	10	11	....	.....	3	.....	4	29	9	2.6	20.3	70.4%
8. City Deep, So. Africa, 1800-lb. stamps, 3-mesh screen	....	....	....	....	....	.....	.....	.....	.....	.....	19.7	4.9	17.3	69.4%
9. Nissen single-stamp, 2100-lb. stamp, all conditions as preceding	....	....	....	....	....	.....	.....	.....	.....	.....	26.0	5.4	17.2	86.9%
(II) *Steam Stamps:*														
10. Butte practice	19	18	....	17	....	6	.....	40	.....	.....	200	100	18.8	37.6%
11. Tremain	....	....	28	4	11	8	.....	49	.....	.....	12	12	21.5	21.5%
(III) *Chilean Mills:*														
12. S. Dakota, fast, 3-roller and rolls	....	14		18	....	13	.....	.....	16	39	105.6	45	.....	55.7%
13. Santa Elena (Mex.), slow, 2-roller, 8-ft. mill-screen discharge	....	....	....	1	6	.....	.....	.....	26	67	28	15	.....	49.1%
14. La Union (Mex.), slow, 2-roller, 8-ft. mill, overflow discharge	....	....	....	....	....	2	.....	13	5	80	22	13	.....	46.1%
15. El Bote (Mex.), slow, 2-roller, 8-ft. mill, hard dense ore	....	....	....	....	....	.....	.....	.....	30	70	19	13	.....	38.8%
16. Lane Mill, 7 ft. diam., 4 wheels, 42 × 5 in. ea., 10 r.p.m., 1 in. feed	....	....	....	2	4	8	.....	86	.....	.....	25	10	24.2	60.5%
17. Russia, slow speed, 14 to 16 r.p.m., 8′6″ mill-rollers, 6′×12″	....	....	....	....	....	.....	.....	.....	.....	.....	39	16.3	26.5	63.3%
18. Goldfield Cons., fast, 3-roller, 6-ft. Trent — feed	20	46			....	34								
18. Goldfield Cons., fast, 3-roller, 6-ft. Trent — disch.	....	4	50		....	25	.....	6	6	48	75	35	8	17.0%
(IV) *Huntington Mills:*														
19. 6-ft. heavy type — feed	2	56	....	38	....	.....	.....	4						
19. 6-ft. heavy type — disch.	....	7	....	53	....	.....	17	4	5	14	75	12	3.8	23.8%
20. 30-in. Griffin Mill, 200 r.p.m., 1-in. feed	....	....	2	3	....	11	.....	12	4	68	31	20	26.3	40.8%
(V) *Grinding Pans:**														
21. 5-ft. Forwood-Down type of Wheeler pans, 62 r.p.m., roasted ore — feed	....	....	4	9	....	12	.....	5	7					
21. 5-ft. Forwood-Down type of Wheeler pans, 62 r.p.m., roasted ore — disch.	....	....	....	....	....	4	.....	6	12	79	47.5	4.5	1.4	8.1%
22. 5-ft. Forwood-Down type of Wheeler pans, 50 r.p.m., roasted ore — feed	....	....	....	33	16	5	.....	16	30					
22. 5-ft. Forwood-Down type of Wheeler pans, 50 r.p.m., roasted ore — disch.	....	....	....	....	2	6	.....	4	87		17	5	2.6	8.8%
23. 5-ft. Forwood-Down type of Wheeler pans, 57 r.p.m. on raw ore, cost 55¢ per ton. In competition with tube mill, see No. 34, this table — feed	....	....	39	17	....	13	.....	4	27					
23. 5-ft. Forwood-Down type of Wheeler pans, 57 r.p.m. on raw ore, cost 55¢ per ton. In competition with tube mill, see No. 34, this table — disch.	....	....	....	....	....	1	.....	4	95		13	6.5	4.4	8.86%
(VI) *Ball-Mills:*														
24. 89×46 in. mill, 3-in. feed	....	....	11	20	8	12	2	2	44	.....	40	18	22.6	64.6%
25. 106×54 in. mill, 3-in. feed	....	....	16	16	10	6	4	4	44	.....	100	52	22.3	†52.0%
26. 106×54 in. mill, 1½-in. feed	....	....	4	16	15	8	.....	8	6	50	90	52	24.6	†42.5%

(VII) *Tube-Mills:*														
27. 5½×22 ft., 30 r.p.m., S. African Banket	feed	28	16	14	20	15	.....	2	.....	2	4	290‡	100	5.3	17.4%
	disch.	....	....	7	22	29	.....	6	.....	7	27				
28. 5× 22 ft., 28 r.p.m., Goldfield Cons	feed	8	....	13	12	....	41	.....	11	7	7	100	60	7.0	11.6%
	disch.	....	....	.....	.....	7	15	.....	13	20	55				
29. 4×18 ft. Krupp, 31 r.pm.	feed	....	....	.....	5	....	20	.....	22	20	33	104	33	0.9	3.0%
	disch.	....	....	.....	.....	....	6	.....	14	15	65				
30. 5×20 ft. Denver, Eng. Wks. Pachuca, Mex.	feed	....	....	.....	5	....	21	.....	12	20	33	103	51	1.4	2.8%
	disch.	....	....	.....	.....	....	7	.....	13	14	65				
											(−150)				
31. 5×22 ft. Davidsen	feed	....	5	10	16	....	14	.....	12	43		77	50	2.1	3.2%
	disch.	....	....	....	1	....	7	.....	18	73					
32. 4½×20 ft. Abbé	feed	....	....	11	20	....	33	26	10		.....	80	43	1.8	3.4%
	disch.	....	....	.....	5	....	22	22	51						
33. 4½×20 ft. Abbé	feed	....	3	16	16	15	10	2	39		.....	75	30	2.6	6.5%
	disch.	....	....	3	4	10	16	2	64						
34. 3′3″ × 13′3″ Krupp, 5920 lb., pebbles. 31 r.p.m., cost 61.4¢ per ton.															
In competition with grinding pan, see No. 23, this table	feed	....	....	11	15	....	37	18	19		.....	29.1	20	4.7	6.85%
	disch.	....	....	.....	.....	.....	.....	4	96						
35. 8 ft., Hardinge Conical	feed	....	15	36	10	11	6	1	21		.....	75	15	1.5	4.5%
	disch.	....	4	21	14	18	11	2	30						
36. 6×12 ft. Hardinge Conical	feed	25	38	12	7	5	2	.....	10	.....	.....	72	15	6.8	32.6%(?)
	disch.	....	....	.....	1	5	2	.....	91						

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(III) *Chilean Mills:*
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(V) *Grinding Pans:*
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25. M. W. von Bernewitz, Min. & Sci. Press, Vol. 103, page 68.
26. Robt. Allen, Jour. Ch. of Mines, West Australia, Min. Rep., Vol. 54, p. 37.

(VII) *Tube-Mills:*
27. H. Stadtler, loc. cit.
28. No. 21 Tube-mill table, p. 101.
29. No. 14 Tube-mill table, p. 101.
30. No. 17 Tube-mill table, p. 101.
31. No. 4 Tube-mill table, p. 101.
32. No. 5 Tube-mill table, p. 101.
33. No. 19 Tube-mill table, p. 101.
34. G. W. Williams, Eng. & Min. Jour., Vol. 85, p. 348.
35. No. 20 Tube-mill table, p. 101.
36. No. 8 Tube-mill table, p. 101.

* A later note by Ralph Stokes in Min. Wld., Vol. 26, page 5, would make it appear that a change in practice has raised the efficiency to 14.8 per cent.
† Based upon an allowance of 1 h-p. to each ten tons reduced from 3 in. to 1 in. ‡ In addition 6¾ tons coarse ore fed per 24 hours.

NOTE. While appreciating the simplicity and ingenuity of Stadtler's method, I must confess that the apparently low efficiency of the two fine-grinding devices, pans and tube-mills, casts some doubt upon the justice of the comparison for these machines.

(2) **Suitability of Product for Purposes in Hand.** This is a less serious question for either amalgamation or cyanide treatment than for concentration, in which the minimum comminution that will liberate the mineral is the desired end.

From the point of view of the amalgamator, the stamp mill, the slow-speed Chilean mill, and the Huntington mill would stand about in the order named.

For cyaniding, where the ore must be finely crushed to permit satisfactory extraction, the stamp mill, the ball-mill, and the Huntington mill can hardly be expected to furnish a sufficiently finely-ground product when operating at reasonable efficiency. The tube-mill, the slow-speed Chilean, the grinding-pan, and the high-speed Chilean mill would appear to take rank, so far as popularity is concerned, in the order named. The tube-mill is always used following stamps or other suitable primary crushing machines; this also applies to the use of grinding pans. The Chilean mill appears, from the data available at present, to cover a wider range of comminution at high efficiency than any other machine on the market. It is not as yet, however, being extensively used as an ultimate fine grinder.

(3) **Wear-and-tear on Various Parts and Loss of Time Incident Thereto.** On this basis, the stamp mill for coarse-crushing, the ball-mill in its wide range, and the tube-mill in its field of final reduction seem to stand preëminent. The slow-speed Chilean should perhaps be included in this class.

The high-speed Chilean and the Huntington mill, together with their variants, are constantly exposed to the criticism of not being foolproof, and of delaying the rest of the plant due to shut-down, or else of requiring a duplicate or spare equipment if continuous operation is to be ensured.

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*** Especially Desirable for Students' Use**

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PART II

CYANIDING

CHAPTER VIII

HISTORY AND CHEMISTRY OF CYANIDING

History

From the majority of gold and silver ores, when properly prepared, the precious metals may be dissolved by an extremely dilute solution of a cyanide salt. It then remains to separate the gold and silver-bearing, or "pregnant," solution, as it is called, from the ore and to precipitate the metals. This last is readily accomplished by the action of zinc.

This process, so extremely simple and easily comprehended in principle, is at times rendered exceedingly complex by mechanical difficulties and chemical interferences.

It is strange, nevertheless, that the long-known fact, that gold is soluble in cyanide solution, should have remained so long unutilized.

The discussion of the cyanide process of gold and silver extraction begins, naturally, with the history of the cyanides themselves. Authentic information in regard to cyanogen compounds dates from the accidental discovery of prussian blue in 1704 by Diesbach, a German chemist interested in the dyeing industry. Owing, however, to the extremely poisonous nature of many cyanides, and to the ease with which they are prepared from organic material, it is more than likely that many of the poisons which ancient and medieval intriguery prized so highly were, in part at least, cyanide compounds. In any event, Diesbach was the first to place the cyanides upon an investigational basis, and from his work Macquer and Guyton, in France, and Bergman and Scheele, in Germany, were led to important discoveries. Scheele, in 1782, succeeded in isolating prussic acid (HCN) and found that gold, silver, and copper were precipitated by cyanide solution and that the precipitates were soluble in an excess of the reagent. This, however, is not a recognition of the fact that gold and silver are themselves soluble in cyanide solutions. Hagen, another German, is credited with having announced in 1805 or 1806 that gold is soluble in

a solution of prussiate of potash, as potassium cyanide was then called.

Ten years later the brilliant Frenchman Gay-Lussac announced the discovery of a compound, having in many respects the properties of a single element, and gave its composition as carbon and nitrogen in equal units. He proposed the name of 'cyanogen' for the new radical — from the Greek 'Kyanos' blue, referring to the blue color of the ferrocyanide salt of iron. From this point, the history of the cyanogen radical comprises the investigation of its various compounds and their properties; only those, however, which relate to the application of cyanides to the solution of gold and silver will be followed.

In a patent specification covering an electroplating process the Elkington brothers, in 1840, clearly state their recognition of the solubility of gold and silver in cyanide solution, and Bagration, a Russian nobleman, published some researches to the same effect in 1843.

According to Philip Argall * Bagration's conclusions may be summarized as follows:

(1) That cyanide of potassium dissolves metallic gold.

(2) That if the gold is fine it will pass rapidly into solution.

(3) That the electric current does not assist solution.

(4) That heat does assist solution.

(5) That the dissolved gold can be precipitated on metallic surfaces without the aid of electricity.

(6) That air has a marked quickening effect on the solution of gold in cyanide.

All of these generalizations have stood the test of time. Truly a remarkable record, and one that stamps Bagration as one of the fathers of cyanide.

Turning again to the Germans, we find Elsner, in 1844, enunciating the now universally recognized fact that oxygen is essential to the solution of gold and silver in cyanide. This fact was again emphasized by Faraday in 1856. Concerning the equation, expressing the solution of gold in cyanide and known by Elsner's name, there appears to be some doubt as to whether Elsner, Bagration, or the compilers of Watt's Chemical Dictionary are entitled to the credit. This equation is usually written as follows:

$$2\,Au + 4\,KCy + O + H_2O = 2\,AuKCy_2 + 2\,KOH.$$

* Min. & Sci. Press, Vol. 95, page 655.

At this point we will turn to the history of the application of cyanide solutions to the treatment of ores. It would seem highly probable that the discovery of the fact that gold and silver were soluble in cyanide, must have been followed by attempts to apply this principle to the treatment of ores, but if so only one record of such an experiment has been found to date.* In 1846 Dr. Duflos, a German, following Bagration's discovery, tried to treat an arsenical gold ore with potassium ferrocyanide but failed to make an extraction. The investigator considered the use of potassium cyanide but abandoned the idea owing to the fact that the ore carried considerable ferrous oxide, and his experiments had already indicated that ferrocyanide was without action. Thus, did a piece of apparently plausible, *a priori* reasoning ensnare and lead away an investigator, from the threshold of a great discovery. We know of no additional metallurgical researches in this line for almost 20 years until, in 1867, J. H. Rae, an American, appears with a patent process for the treatment of gold and silver ores by means of potassium cyanide aided by the electric current. There is reason to believe that this was tried on a commercial scale, but of this we have no positive proof, and the process appears to have died in its infancy. With the exception of statements by Hahn, in 1870, that gold and silver sulphide are soluble in cyanide, and by Skey, in 1875, warning millmen against the too liberal use of cyanide as an aid to amalgamation, owing to its solvent action on gold, we have nothing more until 1881, when we find Faucett and Sanders, within six months of each other, taking out patents for the use of cyanide as an aid to amalgamation. It is to be noted that up to this point no process has been introduced or patented for the extraction of gold and silver from their ores by the solvent action of cyanide salts unaided by electrolysis.

Now comes Jerome W. Simpson of Newark, New Jersey, in 1885, with a patent entitled a "process for extracting gold, silver, and copper from their ores." In his description he specifies the crushing of the ore, agitation with a solution of potassium cyanide, sodium chloride and ammonium carbonate, settling, and then placing in the supernatant solution, a piece of sheet zinc to precipitate the metals. The solution specified is one with about 3 per cent KCy, 0.2 per cent NH_4CO_3, and 0.1 per cent NaCl. He distinctly states in his patent that he does not wish to be understood

* J. E. Clennell, The Cyanide Hand Book, page 12.

as broadly claiming the exclusive use of potassium cyanide as a solvent, or of zinc as a precipitant. This patent, while covering nearly all the points of present practice, was never followed up, but as the English judge wisely remarked in a decision in regard to the MacArthur-Forrest patent to be discussed later: "if the (MacArthur-Forrest) patent were held valid would not Rae's and Simpson's processes, if now used, be liable to be stopped as infringements? I think they certainly would." Quoting again, Philip Argall says:* "If we rely on the almost unanimous verdict of the courts we must award the palm of discovery to the inventive genius of Rae and of Simpson."

This brings us, naturally, to the MacArthur-Forrest patents of 1887 in England and of 1889 in the United States.† After attempting to cover the entire use of cyanide salts for gold and silver recovery and, as has been indicated, being defeated in this owing to the priority of the Rae and Simpson patents, the MacArthur-Forrest patents had, by 1890, been reduced to the distinctive use of extremely dilute cyanide solution "not exceeding 8 parts of cyanogen in 1000 parts of water ‡ and of filiform or 'shaving' zinc for precipitation." MacArthur and Forrest claim that such dilute solutions have a selective action, such as to dissolve the gold and silver in preference to the baser metals. Eventually, in almost every country the MacArthur-Forrest patents were canceled on grounds of lack of novelty. Argall § points out the entire invalidity of the solution patents in that, in all of them, MacArthur-Forrest limited themselves to the use of cyanide alone, excluding the necessary agent, oxygen.

The fight over the MacArthur-Forrest patents was a long and a bitter one, and in litigation so fierce almost every one interested was forced to take sides pro or con, with the result that it is difficult, although the smoke of the battle has cleared, to find a historian free from prejudice in the matter. This much, however, must be said, that whether the patents of the two valiant Scotsmen and their associates were technically valid or not, their work was quantitative, where that of their predecessors was merely qualitative, so that no one can dispute the statement that, "To these men,

* Min. & Sci. Press, Vol. 95, page 682.

† Louis Janin, Jr., has recently called attention to a caveat granted him in 1886, following the lines afterward covered by MacArthur and Forrest.

‡ Equivalent to 1.9 per cent KCy.

§ Loc. cit.

MacArthur and Forrest, is due the praise, not that they were the first discoverers, but that they pushed their invention and enabled a recovery of $14,000,000 in five years by their process, which otherwise would be irrevocably lost," and "that they made what had before been regarded as a pretty chemical scheme, into a sound commercial success." New Zealand, in 1889, was the first country to see cyaniding commercially successful, but South Africa was the more fertile field for the new process.

In 1890, the first year that MacArthur and Forrest report a production by their process in the Transvaal, they recovered $6000 from tailings and in 1891, $60,000. By 1893 over $6,000,000 in round numbers was the cyanide production from the South African mines.

Since the termination of the litigation referred to, the process has advanced at a rapid rate, mainly, however, by the improvement of existing mechanical devices, or by introducing and adapting to the cyanide process, appliances already in use in other industries.

Chemistry

Cyanogen and Its Compounds. Cyanogen, CN, nitride of carbon, is a colorless gas with a peculiar, characteristic odor, resembling that of bitter almonds. It is extremely poisonous. Cyanogen gas is given off from the dry distillation of silver or mercuric cyanides. Although a compound, it has many of the properties of a single element and in many ways resembles the halogens, iodine, chlorine, and bromine. The compound symbol CN is somewhat awkward and must be bracketed for use with subscripts; for this reason I shall follow the British usage of writing the symbol Cy except where a decomposition of the cyanogen radical is involved. Combined with hydrogen, cyanogen forms prussic or hydrocyanic acid HCy. This is most readily prepared by treating a cyanide salt with a dilute mineral acid.

$$KCy + HCl = KCl + HCy.$$

Hydrocyanic acid is a colorless volatile liquid, which boils at 26° C. and is extremely poisonous. It is soluble in water. With the halogens, cyanogen forms several compounds; of these the chloride CyCl is a gaseous compound, while $CyCl_3$ is a solid. CyBr and $CyBr_3$ are both solids as is also the iodide CyI. These haloid cyanides are important in certain modifications of the cyanide process to be discussed later.

The metallic cyanides and double cyanides are of extreme importance in the process. The cyanides of the alkalis are used as solvents. The importance of the precious metal cyanides is obvious enough. The cyanides of the base metals are unfortunately formed to a greater or less extent during the solution of the gold and silver, and are either precipitated later or remain to foul the solution.

The following table of properties is useful:

Cyanides of alkalis and alkaline earths	Cyanides of other metals
Soluble	Insoluble (except Cd and Hg)
Alkaline	Unite with alkaline cyanides to form soluble double salts.
Extremely poisonous	
Colorless	Usually highly colored.

The cyanides of the alkalis and alkaline earths are the substances which have been used and proposed for use as the solvent salts in the process under discussion. Potassium and sodium cyanides are practically the only ones that have been utilized on a commercial scale. The following table is adapted from Julian and Smart:*

TABLE X

Cyanides	Molecular weights	Equivalent amount per unit gold	Dissolving power in terms of KCy
NH_4Cy	18+26=44	88	148
$NaCy$	23+26=49	98	133
KCy	39+26=65	130	100
$MgCy_2$	24+52=76	76	171
$CaCy_2$	40+52=92	92	141
$SrCy_2$	87.5+52=140	140	93
$BaCy_2$	136.8+52=189	189	70

Owing to instability of the alkaline earth cyanides, only the cyanides of the alkali metals and of ammonium are possible as commercial solvents. Of these, ammonium cyanide is too unstable for present use. The sodium salt, while more powerful than that of potassium and to be preferred on that account, is deliquescent and subject to decomposition. Much, if not most, of the

* Cyaniding Gold and Silver Ores, 2nd Ed., page 104.

so-called '98% potassium cyanide' commonly specified is in reality impure sodium cyanide.

The cyanides of the metals and their properties are important in a process in which complex ores are subjected to cyanide solution. As previously mentioned, the simple cyanides of mercury and cadmium are alone soluble; many of the other metals, however, form simple cyanides which, uniting with the excess of cyanide salt in the solution, become themselves soluble compounds. The three metals which of necessity are the most important in the process, *viz.*, gold and silver as solutes, and zinc as a precipitant, belong to this class.

Cyanides of Gold and Silver. The following compounds are known and have been investigated:*

TABLE XI

Symbol	Name	Properties
$AuCy$	Aurous cyanide	insoluble
$AuCy_3$	Auric cyanide	soluble
$KAuCy_2(AuCy+KCy)$	Auro-potassic cyanide	soluble in 4 pts. cold water
$Zn(AuCy_2)_2$	Auro-zincic cyanide	
$AgAuCy_2$	Auro-argentic cyanide	
$AgCy$	Argento-cyanide	insoluble
$KAgCy_2(KCy+AgCy)$	Argento-potassic cyanide	soluble in 8 pts. cold water

Other metallic cyanides of importance in the treatment of ores, with their properties, are given in the following table:

TABLE XII

Metal	Cyanide	Properties
Copper	Cuprous cyanide Cu_2Cy_2	insoluble
	Cupric cyanide $CuCy_2$	partly soluble
	Double cyanides of copper and ammonium $x(NH_4Cy)y(CuCy_2)z(Cu_2Cy_2)$	soluble, feeble solvents of gold
	Cuprammonium cyanides $x(NH_3)y(CuCy_2)z(Cu_2Cy_2)$	slightly soluble, powerful solvent of gold
Mercury	Mercuric cyanide $HgCy_2$	soluble in 8 pts. cold water
	Double cyanide of mercury and potassium K_2HgCy_4	soluble in 4 pts. cold water
Zinc...	Zinc cyanide $ZnCy_2$	insoluble
	Zinc potassium cyanide K_2ZnCy_4	readily soluble

* Clennell, op. cit., pages 71–81.

Complex cyanogen compounds, in which the radical is composed of a metal of the iron group with cyanogen, are of interest chemically, but are not usually of importance in the treatment of ores. Of this group the ferrocyanides of the type M_4FeCy_6 and the ferricyanides M_3FeCy_6 are the most important and best known.

Cyanogen unites with oxygen in some cases to form an acid radical CNO; such compounds are called cyanates. Other members of the oxygen group sometimes replace the oxygen, for example sulphur in the sulpho- or thio-cyanates, potassium sulphocyanate KCyS being a common laboratory reagent.

Solution of Gold in Cyanide. The general truth of Elsner's equation $2\,Au + 4\,KCy + O + H_2O = 2\,AuKCy_2 + 2\,KOH$, as expressing the results accompanying the solution of gold in cyanide solution, is now almost universally accepted.

"It is doubtful, however, whether the complete chemistry of these changes is understood even by those who have made it a careful study."

The hypothesis offered by Julian and Smart seems the most logical explanation of the matter yet presented. This may be briefly summarized as follows:

The solution of gold in cyanide is effected by electrolytic action, the difference of potential set up being due to other minerals in electrical contact in the same solution with gold, or to gold of different densities being in electrical contact.

The common minerals, *e.g.*, pyrite and arsenopyrite, are electronegative to gold; so that if we take the case of gold associated with pyrite and immersed in cyanide solution, the gold becomes the anode of a simple cell, pyrite being the cathode. A current is established with the result that KCy is ionized to K cations and Cy anions. (Fig. 60.)

At the anode $\quad Au + Cy = AuCy.$

$$AuCy + KCy = AuKCy_2$$

At the cathode $\quad K + H_2O = KOH + H.$

This last explains why oxygen in some form or other is essential to continuous solution, for without it the cathode becomes covered with bubbles of hydrogen (polarized) and no further current can flow. Strictly speaking, therefore, it is the negative pole (pyrite) which requires the oxygen and not the gold as commonly supposed. There is indeed some reason to think that free oxygen retards the

solution of the gold in spite of its indispensable secondary effect. Atmospheric oxygen has proven the most satisfactory depolarizer. Artificial oxidizers have not been wholly successful, due probably to the fact that they add to the density of the solution and so retard the migration of the ions. Reducing agents are, of course, manifest hindrances to the solution of gold and silver in potassium

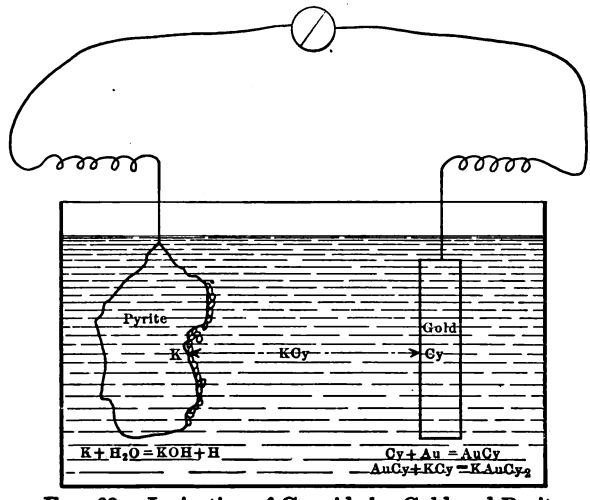

FIG. 60. Ionization of Cyanide by Gold and Pyrite.

cyanide. Experimentally, however, it has been proven that certain compounds in cyanide solution will make it an active solvent of gold and silver in the absence of oxygen.

The use of halogen-cyanides has been mentioned in this connection; according to Professor Christy,* the reaction is as follows:

$$BrCy + KCy = KBr + 2\,Cy \text{ and } 2\,KCy + 2\,Au + 2\,Cy = 2\,KAuCy_2$$

or combining the two:

$$3\,KCy + BrCy + 2\,Au = 2\,AuKCy_2 + KBr.$$

Christy also states that practically the same result is produced by the addition of bromine water to potassium cyanide solution.

$$2\,KCy + Au + Br = KAuCy_2 + KBr.$$

Similar reactions probably hold for chlorine and iodine. Comparing this equation with the Elsner reaction, it will be noted,

* S. B. Christy, Trans. A.I.M.E., Vol. 26, page 740.

that, weight for weight, oxygen causes solution of more gold than any of the halogens.

Double cyanides and copper-ammonium cyanides are likely to break up with a liberation of nascent cyanogen, thus possibly assisting solution in the following manner:

$$Au + KCy + Cy = KAuCy_2.$$

Ferricyanides have at times proven of assistance in dissolving gold and silver.

$$2\,KCy + K_3FeCy_6 + Au = KAuCy_2 + K_4FeCy_6.$$

Direct oxidizers, such as alkaline peroxides, peroxides of the metals, peroxide of hydrogen, ozone, and numerous other compounds, have been tried. Some of them will probably prove efficient under special circumstances, but as a rule their use is to be discouraged.

Precipitation of Gold and Silver. Both gold and silver when in cyanide solution are electro-negative to zinc and hence are precipitated by it. The exact nature of the reaction is not known. We do know, however, that hydrogen is evolved, that KOH is formed, and that *plenty of free cyanide is essential for complete precipitation.* The following reaction has been suggested as meeting all the preceding conditions:

$$KAuCy_2 + 2\,KCy + Zn + H_2O = K_2ZnCy_4 + Au + H + KOH,$$

or, as modified by Christy: *

$$2\,KAuCy_2 + 4\,KCy + 3\,Zn + 2\,H_2O = 2\,Au + 2K_2ZnCy_4 + ZnK_2O_2 + 4\,H.$$

Other reactions consuming both zinc and cyanide doubtless occur without any useful result except perhaps that of furnishing alkali in a most expensive manner. Among these may be mentioned:

$$Zn + 4\,KCy + 2\,H_2O = K_2ZnCy_4 + 2\,KOH + H_2 \quad \text{and}$$
$$Zn + O + H_2O = Zn(OH)_2.$$

The latter compound sometimes forms a white deposit in the zinc boxes, when there is an insufficiency of free cyanide to dissolve it.

Electrolytic precipitation is the term applied to precipitation

* Op. cit., page 764.

where the electric current is supplied from an external source. The chemistry of the process is simply one of ionization.

$$KAuCy_2 + H_2O = Cy_2 + Au + KOH + H.$$

The liberated cyanogen attacks the anodes and the gold is deposited on the cathode.

Countless other precipitation schemes have been tried with more or less success as laboratory methods — such as the use of charcoal, aluminium, sodium amalgam, and various metallic chlorides, nitrates, sulphates, etc.: of these aluminium seems to give the best promise of eventual success and we have at least one recent example of its successful use.* The zinc and the electrolytic precipitation methods, however, are the only ones that are at present in general use.

* E. M. Hamilton, Eng. & Min. Jour., Vol. 95, page 935.

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CHAPTER IX

PREPARATION OF ORES FOR CYANIDE TREATMENT

The preparation of ores for cyanide treatment is dependent not only upon the method of cyaniding to be used but also upon the processes which are to precede cyanide treatment. The conditions governing the choice of methods to be used will be discussed later. In the meantime, certain preliminary processes of importance will be discussed in their relation to cyaniding. The most important of these are: *Crushing*, and *classification* and *dewatering* which are essential in the cyanide treatment of nearly all ores; and *amalgamation*, *concentration*, and *roasting* which are more limited in their application.

Crushing. In crushing for cyanide treatment the aim is, of course, to liberate the gold and silver particles and to expose them to the solvent action of the solution. Anything beyond this is a waste of money and time. It is evident, however, that to fulfill this condition with one ore may require an entirely different degree of crushing than would be necessary with another, as the following extremes will show:

In the Stonewall district, Lower California, I have seen ore crushed by a Dodge rock-breaker and loaded direct into leaching tanks; in other cases tube and ball-mills are used to produce an impalpable slime before cyanide treatment. Where the primary and sole purpose of the crushing is to prepare the ore for cyanide treatment it may be accomplished by either of two methods:

1. *Dry crushing.*
2. *Wet crushing.*

The advantages and disadvantages of each method have formed a fruitful subject of discussion ever since the inception of the process. The chief points of comparison are:

(1) *First-cost of plant per ton capacity.* The cost of rolls or ball-mills is less than that of wet-crushing stamps or Chilean mills.

(2) *Total cost of crushing per ton.* Where ore can be cheaply dried, dry crushing is cheaper, but with wet ore and expensive fuel, wet-crushing will prove more economical.

(3) *Percentage of slime or dust produced.* In general a much more leachable product is secured by dry-crushing than by wet.

(4) *Losses from dust or slime.* This is not an especially serious item in any event but is probably greater in dry-crushing.

(5) *Percolation and leaching of product, and consumption of cyanide.* When an ore is crushed wet and then sized, clean, leachable products are secured, in part at least, and the solvent action of the water used in crushing assists very materially in removal of free acid and so reduces consumption of cyanide.

(6) *Effect upon operatives.* Dry-crushing is exceedingly dangerous to workmen on account of dust, and in time almost invariably leads to fatal lung diseases.

We can never have one general, standard system of crushing, with so many different conditions to meet. An ideal system on a certain ore in one locality might be altogether unsuitable for a similar ore in another locality.

A typical dry-crushing mill would operate somewhat as follows: After reduction to about ½-in. cubes the ore would pass to a dryer, in order that it may be screened and so lessen the slime produced in subsequent comminution. Further reduction may take place either in rolls, as is common in the United States, or in ball-mills which are coming rapidly into favor. Stamps have become obsolete as dry crushers and are seldom or never used. The product from these operations is screened and the over-size returned, preferably to a separate machine. The screened product is ready to be charged into the cyanide leaching-tanks.

Wet crushing is seldom seen dissociated from either amalgamation or concentration or both. Since tube-mills are more efficient when operated wet, however, there is no reason why a non-concentrating, non-amalgamating ore should not, where fine crushing is necessary, be crushed wet by stamps, or ball-mills, or Chilean mills, and go thence after classification to the 'tubes' before cyaniding. Where this system is followed, however, there are few gold ores which would not repay the slight expenses of allowing the tube-mill pulp at least, to flow over amalgamated copper plates.

Classification and Dewatering. The separation of slime from sand, or of excess water or solution from either or both of them, is in the majority of cases necessary at some stage of the cyaniding process.

In essence, classification and dewatering are practically the same, for in cyanide work classifiers with a rising current or hydraulic sorting-column are seldom used, since the problem is always to avoid dilution of the pulp, especially where the object is to prepare sand for tube-mill feed. The classifiers used in cyanide work may be divided into two classes: (1) Cone-classifiers and (2) Drag or conveyor-classifiers.

The *Cone-Classifier*, as its name implies, consists of a hollow cone of wood or preferably steel, the spigot or sand outlet being at the apex of the cone, the slime discharge being provided for by an annular launder or trough about the periphery.

The amount of sand discharged is regulated either by a cock or gate on the spigot, or, as in the Callow tank, by an upturned gooseneck outlet. The feed to the cone-classifiers is usually central, through a pipe dipping a few inches below the level of the pulp in the cone.

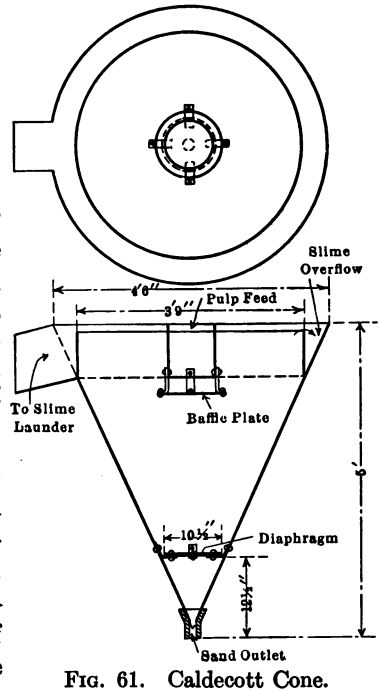

FIG. 61. Caldecott Cone.

There has recently been introduced in South Africa * and in Mexico † the modification suggested by W. A. Caldecott (Fig. 61). This consists of a disc diaphragm placed near the bottom of the cone, together with, in some cases, a baffle plate *B* to prevent 'diving' of the incoming pulp stream. Quite remarkable results are reported as a result of the use of this apparently simple device, a slime-free sand discharge containing only 26 per cent moisture being maintained under working conditions; thus apparently it serves most effectually the dual function of dewatering and classification. Included under cone-classifiers should be mentioned,

* G. O. Smart, Min. Sci., Vol. 61, pages 392-396 from J. C. M. & M. Soc. S. A.

† Walter Neal, Min. & Sci. Press, Vol. 100, pages 483-486.

as of the same general variety, pyramidal boxes of the surface-current box-classifier type, most of which have proven less efficient than the cone for this work.

Drag or Conveyor-Classifiers. The pioneer of this type, designed according to its originator "in desperation * to save mill height," is the invention of J. V. N. Dorr.

The Dorr Classifier, as described by the patentee and manufacturer, and as illustrated in Fig. 62, consists of a settling box in the form of an inclined trough, open at the upper end. From this trough mechanically operated rakes remove the heavy material as fast as it settles. The liquid and slime overflow at the closed end.

The rakes are suspended by suitable hangers from bell-cranks

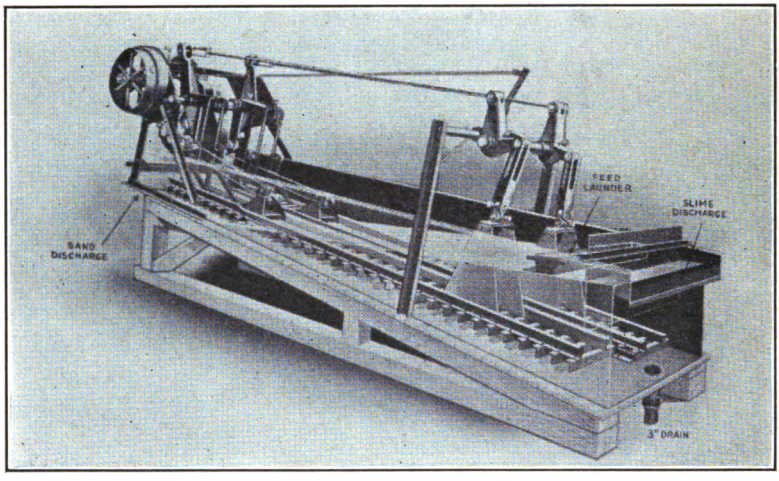

FIG. 62. The Dorr Classifier.

connected by rods to levers which terminate in rollers. The latter press against cams attached to the crank shaft. The rakes are raised and lowered at opposite ends of the stroke by the action of the cams transmitted through the levers and rods to the bell cranks. The horizontal motion is produced by cranks. The pulp is fed across the width of the trough and the sand settles to the bottom, after which it is advanced up the inclined bottom of the trough by the scrapers reciprocating with a slow raking motion. After emerging from the liquid the sand passes across a washing

* Min. & Min., Vol. 28, page 541.

device and is discharged from the open end of the machine with about 26 per cent moisture. It is claimed that if the machine be stopped while full of sand, the rakes can be raised from the bottom and, after it has been started, they can be gradually lowered while running until they assume their normal position.

The slime is prevented from settling by the flow of liquid, as well as by the agitation near the bottom, caused by the reciprocating motion of the scrapers, and overflows at the lower end of the machine. The agitation produced, while ample to prevent the slime from settling, is not sufficient to cause any sand to overflow with the slime.

The Akins Classifier of the Colorado Iron Works, Fig. 63, has a semi-cylindrical trough in which a large slow-moving screw-con-

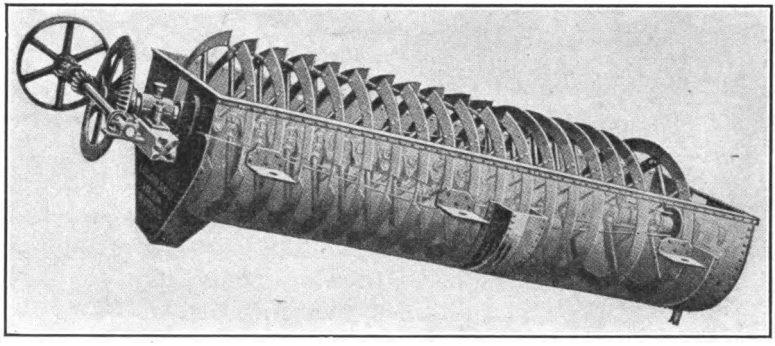

FIG. 63. Akins Classifier.

veyor moves the sand up the slope of the trough, the slime overflowing at the lower end.

Both of the above machines are reported as doing good work.

Amalgamation and Concentration. Amalgamation and concentration are alike, in that both of them are used to save gold or silver not readily recoverable by direct cyanide treatment. The first has been sufficiently discussed elsewhere in this book.

Concentration, where it is practiced in connection with cyaniding, is hardly the fine art that it has become in the treatment of lead and copper ores. In general it comes under what is usually designated as 'fine concentration,' *i.e.*, only finely crushed material is handled. This limits the machines to be used for the work, so that Wilfley tables or other machines of the riffled-surface shaking-table type, with belt vanners occupying a less prominent

position, are almost the only concentrators ever seen in cyanide plants. Canvas or other stationary sloping-plane tables are sometimes used. An interesting variant from this practice is shown in the Goldfield Consolidated plant, where blankets have taken the place of part of the amalgamating tables and are found efficient in recovering coarse but unamalgamable gold.

Roasting. The vast majority of complex ores would be rendered more susceptible to cyanide treatment by a thorough preliminary roast, not only on account of the removal of cyanicides, but also, because of the porosity induced by subjecting them to heat and consequent dehydration. For example, clay before baking is anything but porous, but after being burnt into bricks it will absorb its own weight of water almost instantly.

The extra cost entailed by roasting is, however, seldom justified and only in a few cases, such as in the treatment of the sulphotelluride ores of Cripple Creek and Kalgoorlie, or the complex arsenical-sulphides of Mercur, has it been used on any large scale.* Small installations for the roasting of concentrates prior to cyaniding have been more common.

After numerous trials of the almost endless variety of roasting furnaces which have been evolved in lead and copper metallurgy, standard practice of to-day has fixed upon the straight-line type of furnace as the most suitable for roasting preliminary to cyaniding. One thing is to be particularly noted in this connection: the roasting must be a 'dead' or 'sweet' roast, that is, the elimination of sulphur, arsenic, and other volatile elements must be practically complete. Partial decomposition products of oxidation by roasting are quite as vigorous cyanicides as the same or similar compounds produced by natural oxidation.

There are on the market at present two furnaces which have proven particularly adapted to this work, for both of which we are indebted to Western Australia. I refer to the Edwards and the Merton furnaces.

The Edwards Roaster is of two types, fixed and tilting. In the former the hearth slopes gently toward the discharge end; in the tilting type this slope is adjustable by jack-screws at one end of the furnace, the entire structure being mounted upon a pivot near

* The roasting plant of the Mercur type, erected at Republic, was, in the state of the art at that time, justified by the close-grained character of the ore and not because of the complexity of the gold and silver minerals.

its center, see Fig. 64. Rabbling or stirring of the charge is provided for by means of rotating arms mounted upon spindles projecting through the arch of the furnace and driven by suitable gearing. These rabble-arms cut each others' circles, and revolve in opposite directions, so that the ore is made to pursue a zig-zag course in traveling through the furnace. The standard simplex furnace of this type has a hearth $6\frac{1}{2}$ by 57 ft. In the 'duplex' furnace the hearth is double this width, 13 ft., and has a double row of rabbles; the length is variable, depending upon conditions. The number and location of the fireboxes is a matter determined by the character of the ore. The furnace may be used with either wood, coal, gas, or oil. The furnaces, $6\frac{1}{2}$ by 64 ft., at the Great Boulder mine have a capacity of 25 tons of ore per day, require 2 h.-p., and reduce the sulphur from 4 per cent to 0.05 per cent S and 1.8 per cent SO_3. Fuel required (wood at $2.35 per ton) is 9 per cent of the weight of ore roasted. The total cost of roasting, exclusive of transporting the ore to and from the furnace, is in the neighborhood of 50 cents per ton.

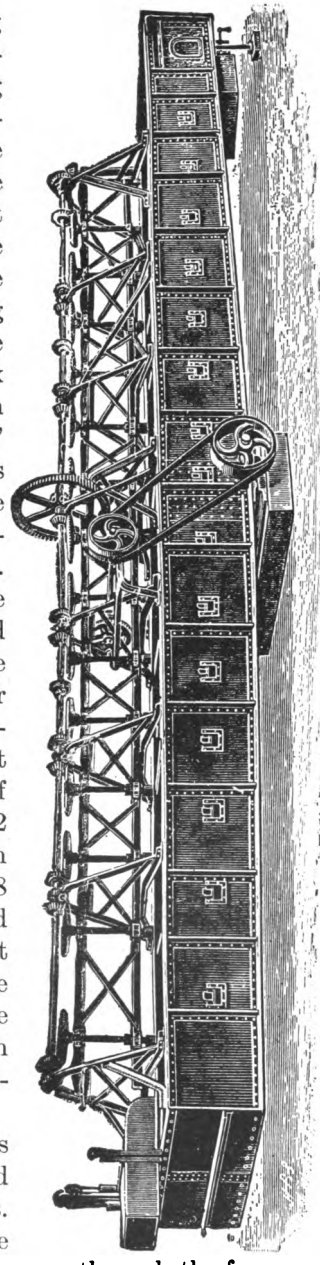

FIG. 64. Edwards Tilting Roasting Furnace.

The Merton Furnace, Fig. 65, is similar in general principle and rabbling device to the Edwards. It is built, however, with multiple hearths, the ore dropping in its progress through the furnace from

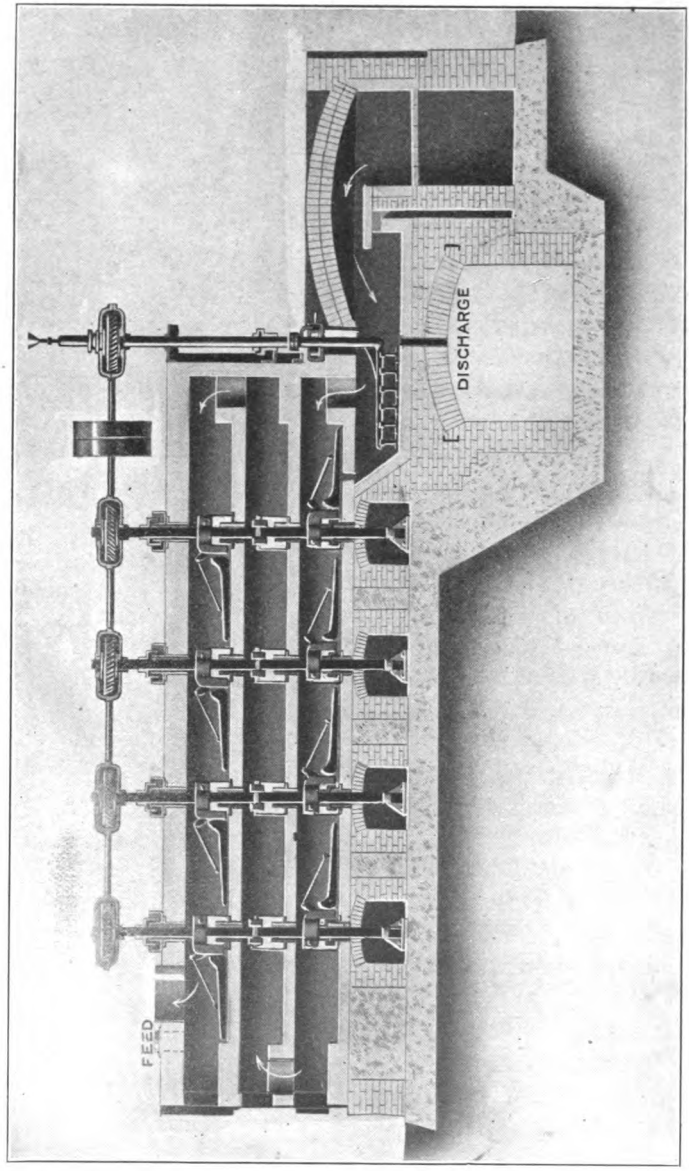

FIG. 65. Longitudinal Section Merton Roasting Furnace.

one hearth to the other. The multiple-hearth feature effects a saving of fuel. The costs and results, otherwise, are about the same as with the Edwards.

Roasting furnaces are also used to some extent as a means of chloridizing sulphide silver ores preliminary to cyanide treatment.

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*** Especially Desirable for Students' Use**

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CHAPTER X

DISSOLVING THE GOLD AND SILVER

The chemistry of solution has already been touched upon; in addition to the reactions involved, the following points are of importance in dissolving gold and silver from ores:

(1) *Strength of the solution.*

(2) *Temperature of the solution.*

(3) *Condition of the gold or silver particles.*

(4) *Presence of cyanicides.*

(5) *Amount of solution.*

(1) **Strength of Solution.** Under ordinary conditions of temperature and pressure the solubility of gold and silver is dependent, according to Julian and Smart,* upon the following circumstances:

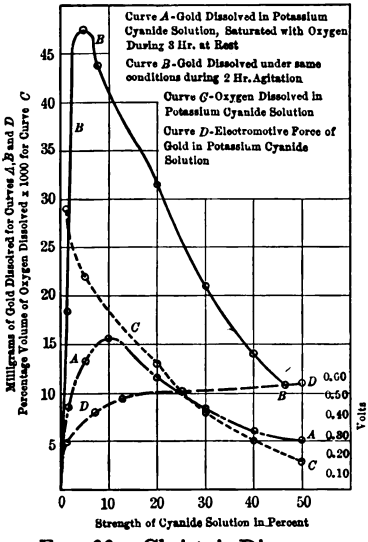

FIG. 66. Christy's Diagram.

(1) The number and velocity of the migratory ions and oxygen molecules that take part in the dissolving of the metals.

(2) The difference of potential between two parts of the metallic surface in contact with the solution.

Christy, whose work upon the electromotive force of metals in cyanide solutions † has become one of the few classics of cyanide literature, states it somewhat differently. According to his view the solubility of gold is dependent upon:

(1) The electromotive force of the gold itself, which is insufficient for the purpose.

* Cyaniding Gold and Silver Ores, 1904, page 77.

† Trans. A.I.M.E., Vol. 30, page 864–946.

(2) The electromotive force of the hydroxyl (OH) ions, which he assumes as proportional to the number of oxygen molecules, *i.e.*, to the solubility of the oxygen in the cyanide solution.

In effect the two authorities say the same thing in a different way.

The curves in Fig. 66, taken from Christy (*op. cit.*), and based in part upon the pioneer work of J. S. Maclaurin,* regarding the effect of cyanide solutions of varying strength with and without oxygen, show clearly that the *maximum solubility* of gold occurs at a concentration of between 5 and 10 per cent KCy, that is where the gold voltage and the solubility of oxygen taken together form a maximum in effect.

This, however, is not the same thing as the *maximum efficiency* of the solution, as is illustrated by the following table of comparative solubilities:†

TABLE XIII

COMPARATIVE WEIGHTS DISSOLVED FROM SURFACES OF EQUAL AREA IN THE SAME TIME AND UNDER IDENTICAL CONDITIONS

Metal	1 per cent sol. KCy	0.10 per cent sol. KCy
Zinc	282	100
Gold	230	82.5
Silver	127	45.5
Lead	121	43
Chalcopyrite	101	36
Charcoal iron	62	22
Pyrite and cast iron	50	18
Galena	32	11.5
Iron rust	20	7
Mispickel	13	4.8
Marcasite	0	0.0

This table illustrates, also, that the dissolving power is not directly proportional to the strength of the solution. For example, under the conditions of the above tests, 230 parts of gold were dissolved in a 1 per cent solution; but in a solution $\frac{1}{10}$ as strong instead of $\frac{230}{10} = 23$ parts, we have $82\frac{1}{2}$ parts, or almost four times this. If we take 0.01 as a unit of solution strength and compare the amounts dissolved per unit of strength in any solution we find:

* Jour. Chem. Soc., Vol. 63, page 731, Vols. 67 and 68, pages 199–212.

† Adopted from Julian and Smart, op. cit., page 81.

Strength of solution	Total gold dissolved	Gold dissolved per unit
1.0	230	2.3
0.1	82.5	8.25
0.01	24.0	24.0

Christy found that solution of gold in cyanide practically ceases at 0.001 per cent KCy; hence it follows, of course, that the maximum efficiency lies between this point and Maclaurin's maximum solubility point. Weaker solutions are more efficient due to the relatively larger number of free ions, and, also, because they not only hold oxygen in solution, but, being less viscous, they do not retard the migration of the oxygen (or hydroxyl) ions.

The solution required, then, is one of maximum strength consistent with minimum resistance to the circulation of the oxygen molecules.

Time is, of course, an all important factor in the question of solution, but when, as in the case of a gold ore, solution takes place with considerable rapidity, while washing can only be accomplished slowly, we may virtually disregard the time factor, and the most efficient solution for gold ore is thus usually found to be from 0.05 to 0.10 per cent KCy. In the treatment of silver ores, however, stronger solutions of 0.25 to 0.50 per cent KCy are found to give the best results.

(2) **Temperature of Solutions.** In practice, after many repeated trials, it has been found that only in rare cases does any advantage result from heating the solution. The solubility of the gold and silver increases from 90 units at 0° C. to 146 at 100° C.; but the solubility of pyrite and other mineral matter increases much more rapidly, owing to the decrease in polarization, and so the advantage of increased solvent power is likely to be lost by increased consumption. Only in those climates where the solution is likely to freeze has it been customary to make arrangement for heating it. The impression is gaining ground, however, that solutions when warm are more easily handled in filters, settling tanks, and thickeners, and the practice is extending among the silver mills with some rapidity.

(3) **Condition of the Gold Particles.** The shapes and sizes of the gold particles in any ore must be numerous; but in any typical ore the chances are that the majority of particles will conform to the same general shape, owing to the fact that they were

probably formed at the same time and place, and under identical conditions.

Take the two extreme cases:

(1) The thin, flat plate or flaky gold, in which the surface exposed is practically constant until solution is complete.

(2) The perfect sphere, in which the surface exposed to solution decreases as the action progresses.

The first case calls for no further discussion; the second explains very clearly, why, in many ores, the last traces of gold are so hard to extract.

If we consider a mixture of variously sized spheres the case is even more apparent; the small spheres are completely dissolved before the large ones, the remaining portions of which linger on, dissolving at a constantly decreasing rate.

The actual condition in most ores probably lies between these two extreme cases; few of the gold particles are likely to be true spheres, and hence as they approach dissolution, they tend to become slightly flattened.

(4) **Presence of Cyanicides.** *Cyanicides* are substances destructive of cyanide, that is decomposing it and becoming themselves cyanides or derivatives of cyanide, without yielding any compounds desirable in the process.

Mineral acids are — in strong solutions — absolutely destructive, breaking up even the cyanogen molecule; in weak solutions hydrocyanic acid is evolved.

Wood-fiber, cocoanut-fiber, jute, and similar vegetable tissues destroy cyanide in absence of alkali, when new, but the reaction soon ceases.

Iron or steel decomposes cyanides but slowly. They consume much cyanide, however, as shown by the equation:

$$Fe + 6\,KCy + 2\,H_2O = K_4FeCy_6 + 2\,KOH + 2\,H.$$

Their introduction is unavoidable in the course of crushing operations, due to the attrition of the grinding-machine parts.

Copper may be native in the ore, or in minute amount may come from blasting-caps. It is comparatively negligible, however.

$$2\,Cu + 4\,KCy + 2\,H_2O = K_2Cu_2Cy_4 + 2\,KOH + 2\,H.$$

Lead is extremely soluble in potassium cyanide solution, but it is scarcely possible for it to come in contact with the solution, and so it remains an unimportant factor.

Mercury, an important factor in treatment of amalgamation tailing, consumes cyanide and is precipitated on the zinc in advance of the gold.

Arsenic may occur native in gold ores, but is scarcely acted upon by cyanide.

Tellurium often occurs with telluride compounds in ores; it is slowly attacked by cyanide but, like its sister metal *selenium*, (which is much more soluble) it seldom occurs in such quantity as to make its presence objectionable from the point of view of cyanide consumption.

Pyrite (FeS_2), unoxidized, is practically insoluble in the absence of oxygen. If partial oxidation has taken place before treatment, with formation of $FeSO_4$, then remedial measures must be taken before cyaniding. Pyrite, however, is more apt to oxidize to limonite and metallic sulphur.

Marcasite is less soluble in the presence of oxygen than is pyrite. It is more likely to oxidize to $FeSO_4$, however, with consequently greater cyanide consumption, followed by complicated and undesirable reactions.

(If much $FeSO_4$ is present it may be largely removed by a preliminary leach with water. The balance may be neutralized with alkali which converts it into ferrous hydrate and alkaline sulphate.)

Chalcopyrite, Tetrahedrite, Chalcocite, etc., in fact, any of the copper-iron sulphide compounds, are probably the worst cyanicides we have. Much depends upon the other minerals in the ore. Chalcopyrite, for example, dissolves slowly in potassium cyanide alone. When a more negative mineral, as marcasite, is present solution is much more rapid. The presence of copper is not necessarily fatal to treatment by cyanide. An ore with 2 per cent of copper may be amenable to extraction of its gold content by cyanide and show a profit. On the other hand, an ore containing less than 0.5 per cent copper may not be susceptible of profitable treatment, owing to destruction of cyanide. Much depends, as before stated, upon the mineralogical association in which the copper and gold exist. Copper seems to interfere less in the extracting of gold, than of silver from an ore.

Mispickel (FeAsS) is a common constituent of gold ores. Its direct solubility in cyanide solutions is not great (see diagram). All the arsenic minerals, however, are soluble in alkali and the compounds thus formed seem to exercise a marked reducing action,

thereby interfering seriously at times with the cyaniding of arsenical gold ores and causing a high consumption of cyanide. The decomposition products, hydrated oxides of iron, and arsenic are not especially active. The latter, however, in absence of free alkali decomposes potassium cyanide and liberates hydrocyanic acid. Lime or magnesia should be added to arsenical ores for this reason.

Orpiment and realgar act in a similar way to the arsenic oxides mentioned above.

Stibnite (Sb_2S_3), like arsenic, is sometimes highly injurious. Its decomposing action on cyanide is similar to that of the arsenic compounds, but it is not as readily controlled by addition of alkali.

Sphalerite (ZnS), contrary to expectation, is not an active decomposer of cyanide. Like the copper sulphides, its action seems to depend in a large measure upon the associated minerals in the ore. If partly oxidized its action is much more marked, destroying both cyanide and alkali.

Galena is often associated with gold and silver ores. It is not an especially violent cyanicide, action following only long exposure to solution.

Tellurides usually make an ore difficult to treat, the solvent action being very slow. As in the other cases, much depends upon the association of the minerals. Roasting renders most tellurides easily susceptible of treatment.

Cyanicides Produced by Oxidation. Various partly oxidized products from iron and copper sulphides are active destroyers of cyanide. Ferrous and ferric sulphates unite with cyanide, forming either Prussian or Turnbull's blue. The addition of lime or caustic soda precipitates ferric hydrate and prevents loss of cyanide. Carbonates of copper (malachite and azurite) consume so much cyanide that ores containing even small quantities of them become unprofitable for cyanide treatment.

Mineral Association. Apparently similar ores, from different localities, vary greatly in their action under cyanide treatment. The only explanation which can be offered is that of contact electrolysis varying with the minerals in association. For example, marcasite and pyrite, separately, have comparatively little effect upon cyanide. When the two minerals occur together the destruction of cyanide may be greatly increased.

Many of the complex compounds formed are decomposed in

the precipitating boxes, the metal being deposited or forming insoluble compounds with the precipitate. In this way, apparently weakened solutions are often freed of part of their impurities, and are in a measure regenerated.

(5) **Amount of Solution.** In a thick, slimy pulp, solution of gold in cyanide cannot take place rapidly. This means that a certain ratio of liquid to solid must be maintained for the best work. The ratio varies with different ores and with different degrees of crushing. Silver ores require a greater amount of solution than do gold ores. For ordinary gold-silver ores the ratio will vary from one of solid to one-half of solution in leaching, to as high as one of solid to six of solution in treatment of some silver ores, proportions in each case being by weight.

Method of Application. In the MacArthur-Forrest methods of cyanide treatment, the cyanide solution was passed, usually with the aid of gravity, through a porous mass of ore, dissolving the gold and silver as it went. The cyanide solution was said to 'leach' the ore and the operation was designated the 'leaching method.' It soon came to be realized, however, that in order for the solution to leach out the gold and silver, it must percolate through the mass of ore, and that percolation, in the majority of cases, is only possible when the ore is in a comparatively coarse state; so that leaching is impossible for ores requiring fine crushing to expose the gold particles to the action of cyanide. To overcome this difficulty agitation in one form or another has become a valuable adjunct to the cyanide process.

(A) THE LEACHING METHOD

Whenever it is possible to obtain satisfactory results by it, the leaching method, provided the time required be not too great, is to be preferred on account of its simplicity and consequent low cost.

Leaching-Tanks. The ore after suitable preparation is charged, by means to be discussed later, into the treatment 'tank' or 'vat,' as it is variously called. Leaching-tanks are almost always circular and cylindrical, owing to the greater ease of construction and support when in this shape. Three materials are available, wood, steel, and concrete or masonry.

Wooden Tanks are used in practically all but the largest installations. California redwood, and Oregon or Douglas fir are the

materials most commonly employed. Tanks of redwood are to be preferred. Tanks less than 10 ft. in diameter may be made of 2-in. lumber, but 3-in. material is better. The staves of the tank, when well made, are shaped to the arc of the tank circle and the edges are beveled radially. The bottom pieces are cut to the necessary circle and are secured to each other by wooden dowel pins. The bottom of the tank is notched into the staves to a depth of $\frac{1}{2}$ to $\frac{3}{4}$ in., the notch or gain being known as the *croze*. The staves, of course, project some distance beyond the bottom of the tank, usually about 3 in. This projection is called the *chine*. The staves are secured to the bottom and to each other by encircling rods of $\frac{1}{2}$- to $\frac{3}{4}$-in. round steel. Flat hoops were at one time popular but have been universally condemned. For securing the two ends of a small hoop, or the various sections of a large one,

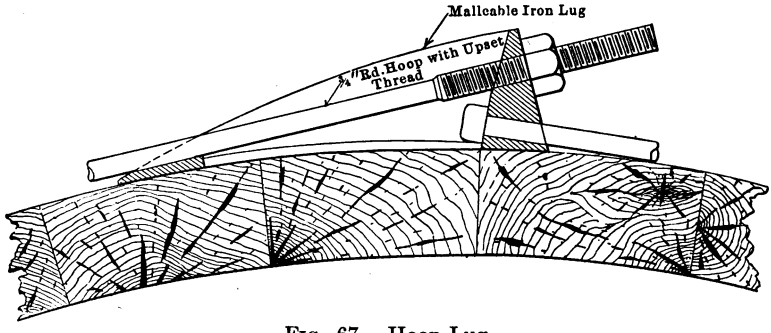

FIG. 67. Hoop-Lug.

to each other, various types of lugs are used. Fig. 67 shows one of the best. The threads of these hoops should be 'upset,' *i.e.*, of the same diameter at the base of the thread as the balance of the rod, thus securing the maximum strength with minimum weight. Wooden tanks are designated by outside diameter and outside height. With 3-in. material, this means that the inside measurements are 6 in. less than the nominal size.

Steel Tanks are popular in South Africa and in the larger installations in Mexico. This is to be explained to some extent by the fact that in a tropical or semi-tropical country wooden tanks are harder to keep from leaking, as well as by the condition that, in the former country at any rate, the deposits justify a permanency of plant which is seldom aimed at in the western United States. They are more expensive than wooden tanks at the present time.

As lumber suitable for tank stock becomes scarcer and higher in price, steel tanks will become more widely used. They are built of the same shape as wooden tanks, the thickness of plate varying from $\frac{3}{16}$ to $\frac{3}{8}$ in. according to height and diameter. They are stiffened at the top and bottom by angle-irons and the bottom corner is usually formed by this means. Like wooden tanks they are shipped 'knocked down,' the plates being cut and punched for rivet-holes, and the side plates being rolled to the required tank circle, as are also the angle-iron rings.

Masonry and concrete tanks have not yet come into extensive use. With the experience of hydraulic engineers in seepage through concrete stand-pipes and reservoirs it would seem, that until more efficient means of water-proofing such materials are devised, masonry and concrete should be reserved for foundations and the like, and wood or steel used for cyanide tanks.

Size of Tanks. Leaching tanks are usually made of such a size that their capacity is equal to the amount of sand to be leached per day. In this way, by providing as many tanks as there are days in the cycle of treatment, one tank is loaded and discharged each day, thus keeping conditions of labor and so forth uniform, throughout the plant.

The depth of the tank to be employed is influenced mainly by the leachability of the ore; from 4 to 10 ft. will cover the range of depths generally employed. Having fixed on the depth, the diameter is a simple matter of arithmetic, as soon as the weight per cubic foot of the material is known. This last may easily be determined by experiment. The commercial size nearest the calculated dimensions should, of course, be selected, making allowance for the fact that the sand should not come nearer than 6 inches from the top of the tank, and also for the space occupied by the filter frame.

Filter-Bottoms (Fig. 68). The filter-frame most commonly used (marked style *B* and style *E*) consists of a grating of 1-in. strips, spaced 1 in. apart, supported on a series of sill-pieces at right angles to them on the bottom of the tank. On the grating is placed a canvas filter-cloth; this in turn is often supported by cocoa-matting. In large tanks this grating is made in sectors for ease in handling. Surrounding this filter-frame is a wooden ring, of the same height as the filter-frame, so set that there is a space of $\frac{3}{4}$ to 1 in. between it and the side of the tank; the canvas filter-cloth

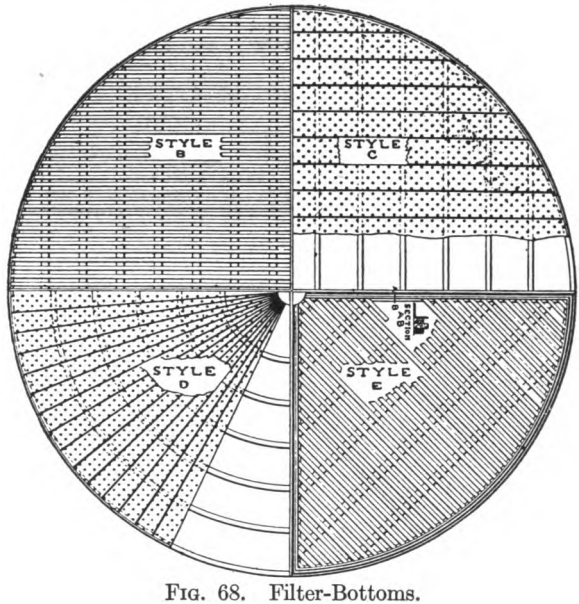

FIG. 68. Filter-Bottoms.

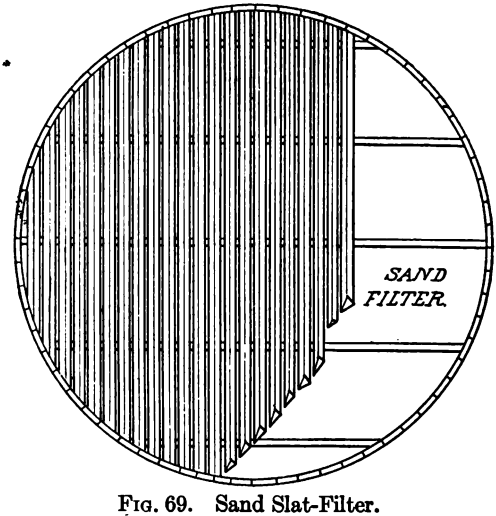

FIG. 69. Sand Slat-Filter.

is cut several inches larger in diameter than the filter-bottom, the edges of the cloth are then tucked into this space, and a rope is caulked on top of the cloth, thus stretching the cloth on the filter-frame much as the vellum is stretched on the head of a drum. In some plants shoveling-strips of 1-in. material are lightly fastened to the upper side of the filter to protect the canvas when discharging the tanks by hand.

Another form of filter (style *C* and style *D*) is made of boards with numerous holes bored through them, the boards in turn being supported on cleats to give space for the filtered solution. So called 'sand-filters' (Fig. 69) are sometimes seen in leaching plants and are becoming quite popular in some mills as a means of clarifying slightly cloudy solutions from other parts of the process. They consist simply of triangular strips, supported on the tank-bottom by the usual cleats. The apex of the triangle is placed uppermost and the space between the slats filled with successive layers of gravel, until a uniform surface of coarse sand is presented. Upon this the charge to be leached is placed.

Charging. The method to be used in charging the leaching-tanks depends primarily upon whether the ore is charged dry or wet. Dry-charging is, of course, only possible with a dry-crushing plant or when dealing with semi-dried tailing accumulations. It has the advantage, that there is no such stratification of coarse and fine material, as necessarily occurs when a wet-crushed product is charged, with water, directly into the leaching tank. In dry-charging for small installations the material is handled by car or other means and is dumped on a grating set over the tank. In larger plants elaborate systems of belt-conveyers either feed to centrifugal-distributors, which scatter the sand in a shower over the surface of the charge, or else pile it in the center of the tank, there to be distributed by a series of discs rotating about a vertical axis, operated and supported from a travelling bridge which spans the leaching tank and travels lengthwise of the tank-house. This distributor is similar in principle to the Blaisdell excavator shown in Fig. 74 and is put out by the same firm.

In wet-charging, two methods are available. The first is to collect the sand in the tank in which it is to be treated. The second is to collect the sand in one tank and transfer it to another before applying the cyanide solution. In small plants the first method is almost always used on account of the smaller num-

ber of tanks required. The pulp is usually charged into the leaching-tank by means of a distributor of the lawn-sprinkler type, (Fig. 70), the slime overflow being provided for, either by a series of holes one above the other in the side of the tank, or by a 'slat

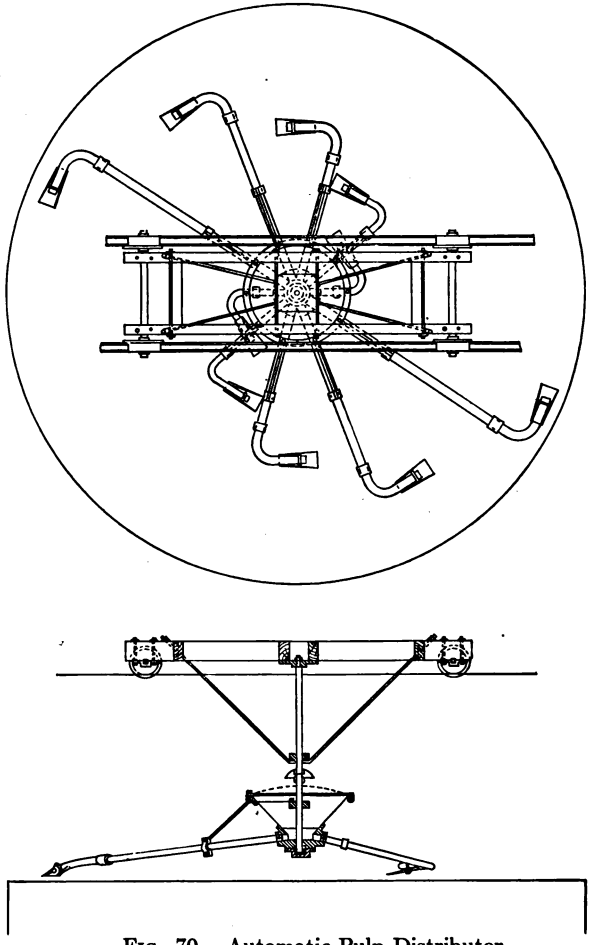

FIG. 70. Automatic Pulp-Distributor.

gate' overflow, so arranged that the slime outlet is at a practically constant distance above the sand accumulating in the tank. By this means, with general mill tailing, the sand is kept fairly free from slime; but slime in the charge and sand in the overflow are, to a certain extent, unavoidable. Usually, however, a fairly leach-

able product is secured. If good classification precedes charging, there is no reason why almost clean sand should not be obtained.

When the tank is sufficiently full, the flow of pulp is diverted, the water on the surface decanted, and that in the charge allowed to drain through the filter. The cyanide solution is then applied.

When collecting tanks are used, they are filled with clear water, and the stream of pulp is turned into the tank through the distributor as before; the sand at once begins to settle, and the slime overflows by means of an annular launder at the top edge of the tank. When the collecting tank is full of sand, the pulp-flow is diverted, the water drained off as before, and the sand transferred

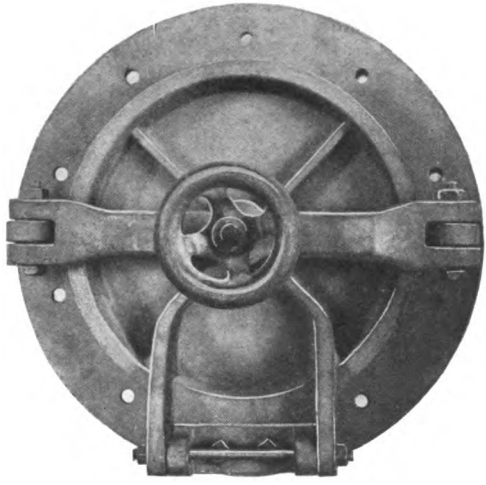

FIG. 71. Bottom Gate.

to the leaching tanks. In some cases it is found advantageous to give the sand a double treatment, that is, to leach it in one tank and then transfer to another tank for a second leaching. The thorough aeration of the pulp thus induced occasionally makes a marked difference in the extraction. In this event the first treatment may be given in the collecting tanks.

Discharging. There are two types of gates for use in discharging the leaching-tanks; one is the bottom gate, the other the side gate. Bottom gates, two forms of which are shown in Figs. 71 and 72, are preferable whenever there is room below the tank to permit their use. These are made either square or circular, and a tube-like projection extends from the bottom of the tank to the

surface of the filter. Circular discharge-doors are to be preferred. Side discharge-gates should be put in only when their use is unavoidable; they interfere seriously with the hoop spacing, unless special provision is made for them. The form shown in Fig. 73

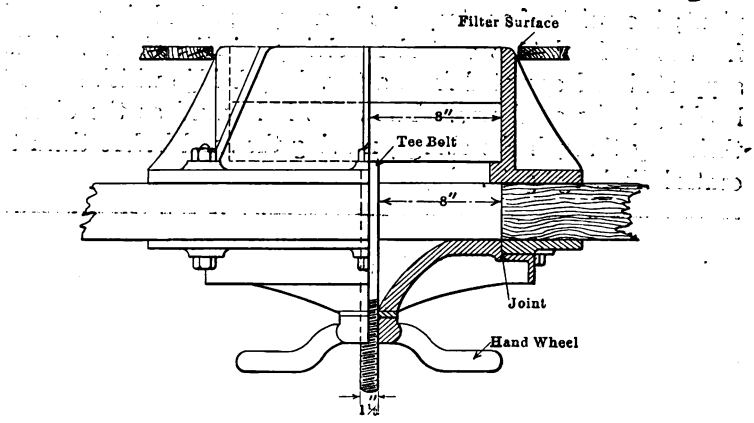

FIG. 72. Bottom Gate.

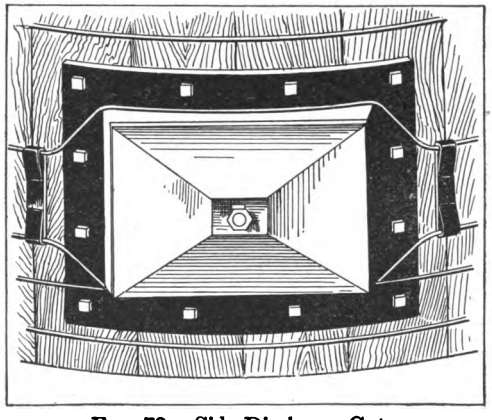

FIG. 73. Side Discharge-Gate.

is recommended by R. Stuart Browne,* who has had much experience in tank construction.

For moving the material out of the tank, there are three possibilities: shoveling by hand, sluicing with a hose and nozzle, and excavating with machinery. Whenever water is available sluicing

* Pac. Miner, Vol. 17, page 336.

is the quickest and cheapest method; shoveling is, of course, extremely expensive, but is often the only alternative where the size of the plant does not warrant the use of mechanical excavators. The Blaisdell excavator (Fig. 74) is only possible with a large plant; it consists of a rotary disc-harrow, the discs being suspended from arms fixed to a central revolving spindle. This spindle is supported and operated from a travelling bridge which spans the leaching-tanks as shown. The shaft is lowered as the discs cut their way through the charge, moving the sand before them to a central discharge-gate.

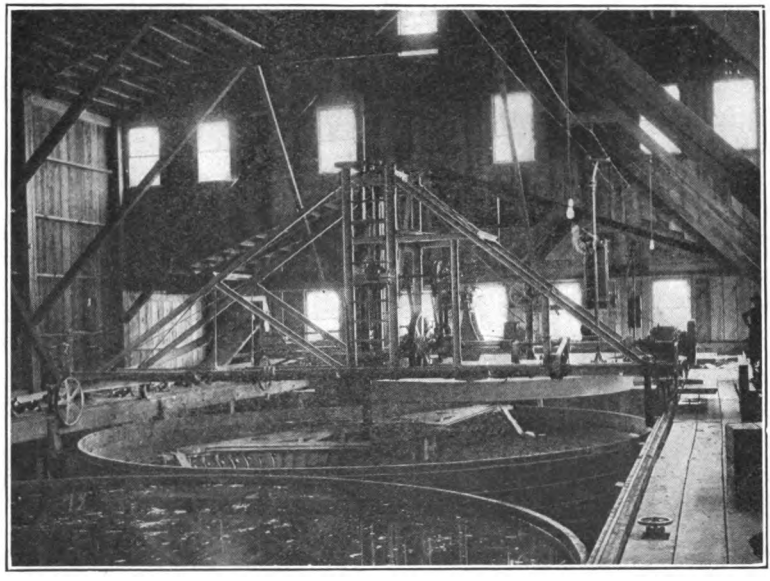

FIG. 74. Blaisdell 30 ft. Class A Vat Excavator.

Applying Solutions. The solution may be applied at the surface of the ore and allowed to percolate downward by gravity, or it may be applied from below the filter-cloth, under low pressure and allowed to percolate slowly upward. There are also the intermittent and the continuous methods of application. In the former, each solution is added after the previous one has completely drained. In the continuous method the next solution is added before its predecessor is out of the tank. There is a diversity of practice in this respect. Some metallurgists apply their solution in a few large charges, other in numerous small ones, both giving

apparently equally satisfactory results. In the subsequent water wash, however, numerous small charges are beneficial, and this is an important point, for it is as essential that the dissolved gold should be washed out of the charge, as that it should be dissolved in the first instance.

The rate of percolation varies with the size, volume, and uniformity of the interstices between the ore particles and to some extent with the depth of the column, the temperature, and the pressure. Downward percolation takes place at a fairly uniform rate, diminishing slightly with increased depth of ore; upward percolation diminishes rapidly in velocity as the surface is reached. Uniformity in texture of the ore is one of the chief secrets of good leaching, and to get uniformity in texture uniformly sized particles are necessary. Thus a carefully-sized ore will give not only more uniform results but better extraction, and will retain less moisture than the same ore unsized.

The amount of moisture retained by an ore depends upon the character of the ore, the size of the particles, the texture, the depth of the column, the temperature, and the pressure. The first 2 to 6 in. above the filter always retains an excess of moisture. This depth is practically constant, so that the deeper the charge in the tank the less moisture proportionally is retained above the filter. Pressure, applied usually by inducing a partial vacuum below the filter, is efficient in removing the last traces of moisture. Pressure, however, must not be applied indiscriminately, as it is apt to result in uneven packing and settling of the charge.

Bad leaching results from uneven percolation, and uneven percolation follows want of uniformity in texture of the ore, or lack of uniformity in the pores of the filter-cloth. If ore containing slime is charged, the slime may settle unevenly on the filter-cloth and so clog part of the pores. For this reason filter-cloths should be frequently washed. In some plants there is an excellent custom of covering the filter with three or four inches of clean sand; this serves to equalize the filtration rate over the whole area. If any machinery or staging is supported directly on the tank, the vibration set up may cause uneven settling, and so lead to poor leaching results.

The quantity of solution required to cover the ore is evidently dependent upon the volume of the interstitial spaces.

If G is the specific gravity of solid ore and g is that of the crushed

material, then the ratio of spaces of voids to volume occupied by pulp is $\frac{G-g}{G}$.

Assume the specific gravity of solid quartz to be 2.65, and of the pulp 1.28, then $\frac{2.65-1.28}{2.65} = \frac{1.37}{2.65} = 0.52$ = proportion of voids. Therefore, the proportion of voids to solids is as 52:48, and the relative weights are: 1.28 of sand to 0.52 of water; or 0.405 ton of water per ton of sand is required to fill the interstitial spaces.

Slime. Reference has already been made to the effect of slime, and it may be well to digress somewhat on this point. The term slime is a somewhat indefinite one; many authorities base the distinction between sand and slime upon fineness of the grains alone. This is manifestly inaccurate, since pure quartz ground to 150 mesh does not form a slime with water, while a talcy, clayey rock will become plastic with far less comminution. The definition proposed by Warwick comes nearer fitting the case than any other I have seen. "A slime is all that part of a crushed product which, after settling in water for 48 hours, retains at least 40 per cent moisture and exhibits plastic qualities." *

Slime is the enemy of leaching, and as it increases in quantity, leaching becomes more and more unsatisfactory and finally impossible. The following taken from Julian and Smart illustrates this point:

To a carefully sampled pulp, freed from its own slime, various quantities of plastic clay, free from gold and silver, were added, and the samples so prepared were each submitted to a leaching test under identical conditions of treatment.

Slime added to sample	Extraction
0%	84.3%
5%	75.9%
10%	62.5%
11%	31.9%
12%	No percolation

The slime was tested for cyanide consumption and showed practically none whatever, so that the effect was purely mechanical,

* This is a working definition. That the sand represents the crystalline portion, and the slime, in the main, the colloidal portion of the pulp is evident.

due in part to coating of the gold particles, and in part to the retention of gold solution by the slime, up to the point where leaching was out of the question.

(B) THE AGITATION METHOD

While much more recent than leaching, agitation has now reached a high state of perfection and the majority of cyanide plants throughout the world are treating non-leachable material by this method. The systems of agitation in use may be divided into:

1. *Agitation with mechanical stirrers.*
2. *Agitation with centrifugal pumps.*
3. *Agitation with compressed air.*

1. Agitation with Mechanical Stirrers. The revolving arm or paddle-agitator (Fig. 75) is one of the simplest of this type. A

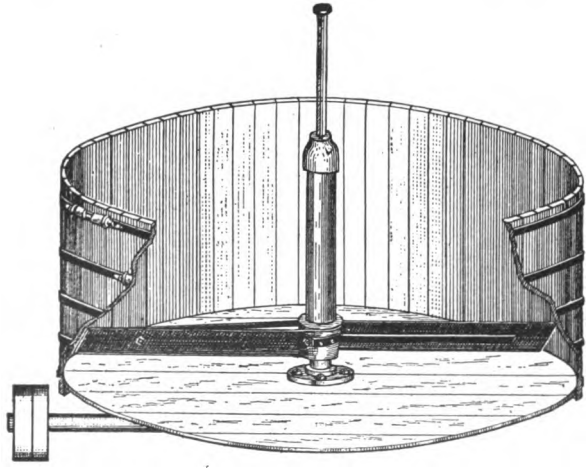

FIG. 75. Paddle-Agitator.

vertical shaft or spindle is supported in the center of a flat-bottom tank. To the lower end of the spindle are attached two horizontal arms, tapered toward the outer end. The shaft and arms are revolved by suitable bevel-gearing above or below the tank. One horizontal shaft may be arranged by suitable clutches to drive any number of agitators.

This agitator does not provide for as thorough aeration of the pulp as is desirable and there is always a tendency for ore and sol-

ution to rotate *en masse*, thus defeating the purpose in view. In consequence it has been in some cases supplemented by baffle-boards, or by perforated pipes on the bottom of the tank through which compressed air is delivered to the charge.

One of the most recent devices for agitating cyanide pulp is the *Dorr agitator* (Fig. 76).

In speaking of it the inventor says:

"It can be operated in any flat-bottom tank, and consists of a central vertical cylinder carried by a shaft supported from the top of the tank and equipped with two arms carrying plows, as in the Dorr thickener, which travel around the bottom of the tank and draw the pulp to the center. This (pulp) is raised through the cylinder by means of air and distributed evenly over the surface of the pulp in the tank, by suitable revolving launders."

It is difficult to see why this apparatus should not prove entirely successful, since it provides for thorough aeration and obviates

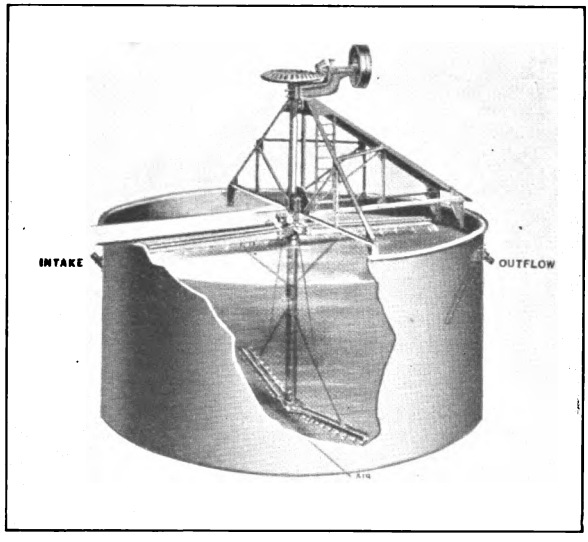

FIG. 76. Dorr Agitator.

the other difficulties of the mechanical-stirrer type of agitator as commonly used. It also makes impossible the accumulation of solids in the bottom of the tank, a defect from which even the best compressed-air agitators are not entirely free, particularly if col-

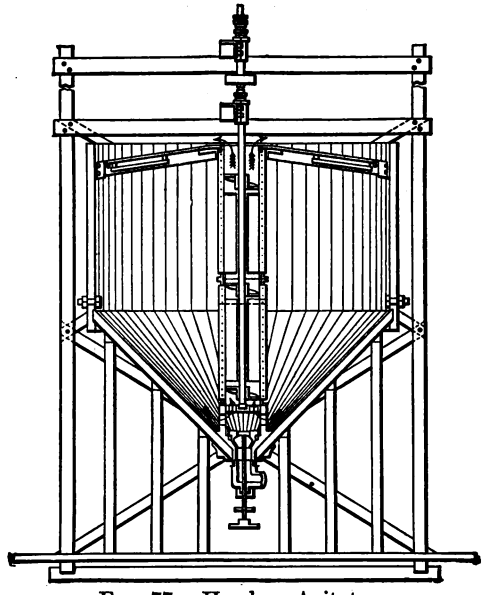

FIG. 77. Hendryx Agitator.

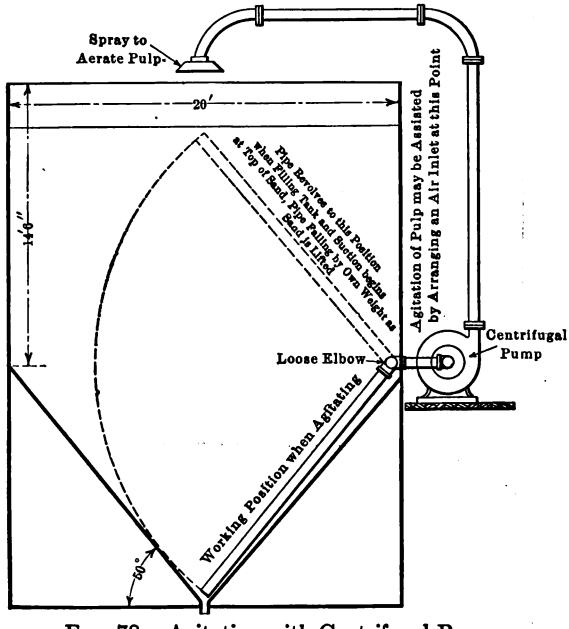
Spray to Aerate Pulp
20′
14′0″
Pipe Revolves to this Position when Filling Tank and Suction begins at Top of Sand, Pipe Falling by Own Weight as Sand is Lifted
Agitation of Pulp may be Assisted by Arranging an Air Inlet at this Point
Centrifugal Pump
Loose Elbow
Working Position when Agitating
50°

FIG. 78. Agitation with Centrifugal Pump.

loids (which assist materially in keeping the sand in suspension) are absent from the ore.

The Hendryx agitator, see Fig. 77, consists of a conical-bottom tank with a central column in which operate several propellers. These propellers draw the pulp from the bottom of the tank into

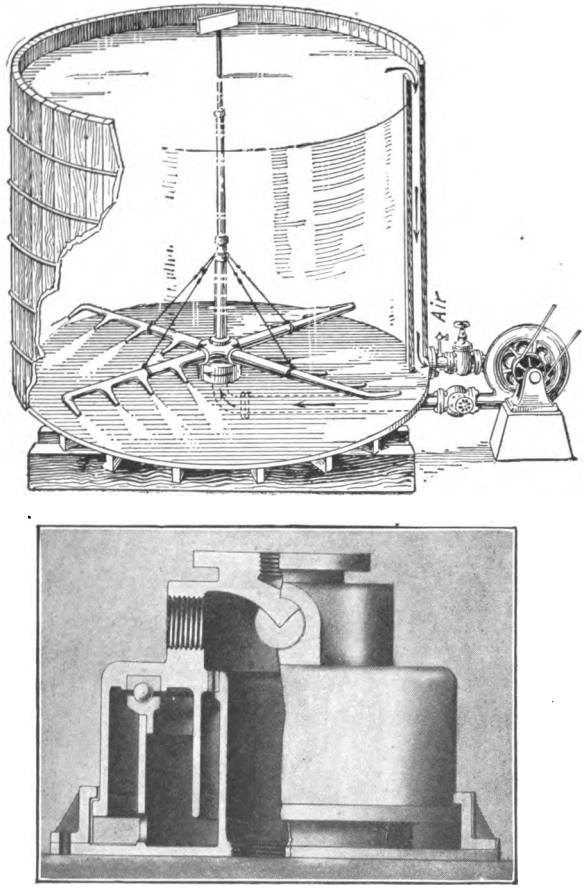

FIG. 79. Trent Agitator with Footstep Detail.

the central pipe, and discharge it at the top on a flat conical apron which extends to within a short distance of the side of the tank. This apron forms an excellent aerating surface.

2. Agitation with Centrifugal Pumps. This system (Fig. 78), which was developed shortly after the first revolving stirrer, consists

of discharging the pulp from the bottom of a conical tank into the suction of a centrifugal pump, returning it to the top of the tank, and discharging it upon an apron, or in the form of a spray, to ensure oxidation. A modification of this method is to allow a small amount of air to be drawn into the pump suction, thus aerating the pulp in the pump and return pipe. This method is open to the objection that the wear of centrifugal pumps when handling

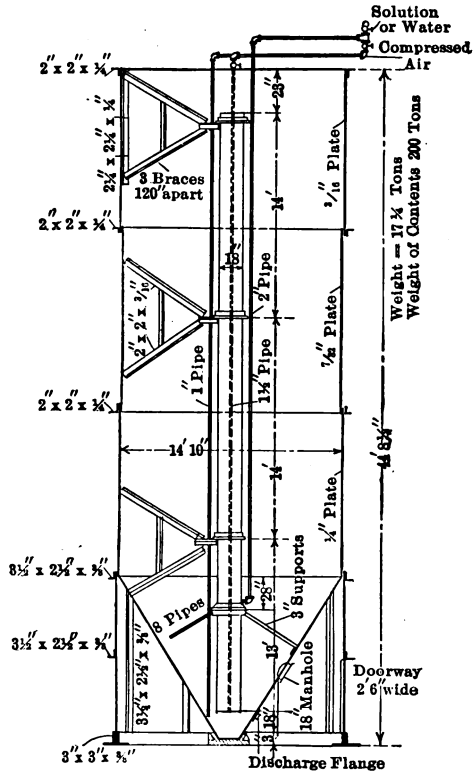

FIG. 80. Pachuca Agitator.

pulp is excessive. To obviate the difficulty, the following scheme has been proposed by L. C. Trent, and has proven fairly successful in several mills:

The Trent agitator (Fig. 79) is, only in a superficial sense, of the centrifugal pump type. The suction of the pump here connects with the upper part of the tank containing practically clear solution; this clear solution is delivered on the bottom of the tank,

through the arms of a distributor of the lawn-sprinkler type, elsewhere discussed. Aeration is provided for by a properly-controlled 'leaky suction.' Special provision must, of course, be made for the footstep bearing carrying the distributor, in order that it may not wear out rapidly, due to the grit.

3. Agitation with Compressed Air. Developed independently in New Zealand and in Mexico, the Brown-Pachuca agitator has rapidly displaced other methods for the circulation and aeration of pulp. The extreme simplicity of the method, combined with the fact that there are no wearing parts exposed to the cutting action of the pulp, has led to its frequent application in recent slime plants.

Fig. 80 shows a typical Pachuca tank, 45 ft. high by 15 ft. in diameter, consisting of a steel shell having a conical bottom. Suitable braces of angle-iron extend from the central pipe-column to the shell. In a tank of the size given the central pipe is 18 in. in diameter. It extends to within 18 in. of the bottom or apex of the cone and runs from there to within 2 ft. of the top of the tank. Into the lower end of this pipe a jet of compressed air is forced through a $1\frac{1}{2}$-in. pipe and the dilution by air bubbles of that part of the pulp within the pipe leads to its rising within the pipe and hence to its over-flow at the top. The following is taken from an article by A. Grothe * the father of the Pachuca tank in Mexico.

"When agitation is started up, a pressure somewhat in excess of the hydrostatic head has to be used, and slime being closely packed in the conical part of the tank and in the air-lift, the first effect of turning on the compressed air will be to drive the contents of the lift upward, where they will overflow into the tank, leaving the air-shaft quite empty. The air-valve should then be closed to obviate the useless escape of air, but in a short time material outside the lift will have entered the lower part of it by reason of the pressure in the tank, and a slight movement of the mass outside the lift will have set in. The contents of the lift should then be blown out again by a momentary opening of the air-valve, and this operation should be repeated until a narrow annular channel is formed on the outside of the lift through which the solution standing above the sand can freely enter the lift at the bottom, after which the current scours the surrounding sand and slime, and after a short time the cone is free from all impediment and the tank is in uniform agitation, the whole operation not having taken more than a few minutes. The pressure and quantity of air can now be considerably reduced; they do not have to be nearly so

* Mex. Min. Jour., Vol. 11, No. 2 (Aug., 1910, page 3).

great as the hydrostatic pressure, corresponding to the height of the tank and the specific gravity of the pulp, owing to the central pipe containing a mixture of air and pulp very much lighter than the material surrounding it when the tank is in agitation.". . .

. . . "It is a common mistake to take for granted that violence of agitation acts beneficially on the time and thoroughness of extraction. It is only necessary to keep all the pulp in constant agitation, and this is attained when the velocity of the current in the cone is sufficient to move every particle of material as it comes down to the inlet of the air-lift. As long as any settlement on the sides of the cone is prevented the maximum of useful agitation is obtained and anything in excess of this is sheer waste of air. The pulp brought to the top instantly commences a downward course, the particles of ore, at a somewhat greater velocity than the solution, being accelerated by gravity. Particles heavier, on account of size or greater specific gravity (such as the metallic contents), reach the bottom sooner than the fine slimes and, therefore, the solid contents continually change their position in relation to the fluid contents, and in their fall of 45 feet are constantly brought into contact with fresh portions of the solution. This is the fundamental difference between this kind of agitation and that in mechanical stirrers, or with ordinary and more refined air jets, all and every one of which have a tendency to effect an appreciable amount of classification, thus producing the opposite effect to that obtained in Pachuca tanks." . . .

. . . "To obtain this perfect circulation, 30 cu. ft. of free air (per minute) is sufficient for a tank 15 ft. diameter and 45 ft. height, holding a charge of 100 tons dry ore, all ground to 150 mesh, and 75 per cent to 80 per cent to 200 mesh, with a proportion of 1.2 of solution to 1 of solids. It is another matter when some peculiarity of the ore, such as a tendency to scum, has to be overcome, in which case it may be judicious to only charge the tank to a point several feet below the top of the air-lift, making use of the falling pulp to break up the scum which might otherwise accumulate and cause trouble. An instance of the successful use of this method is to be found in the San Rafael mill at Pachuca."

In a recent modification the central pipe terminates some 6 or 8 ft. below the top of the charge and better aeration is claimed for this arrangement.

*The Parral Tank,** Fig. 81, is fundamentally the same as the Pachuca. As at present used the Parral tanks are flat-bottomed, 25 ft. in diameter by 15 to 25 ft. high; and the single 18-in. central pipe of the Pachuca is replaced by four 12-in. pipes equidistantly spaced on a 17-ft. circle. These pipes terminate in a

* Bernard Macdonald, Trans. A.I.M.E., Vol. 42, page 819.

tee at their upper end, so that the pulp is discharged with considerable tangential velocity, soon setting the entire contents of the tank in rotation.

Another variation of the compressed-air agitation is the so-called Just process, in which the bottom of a flat treatment tank

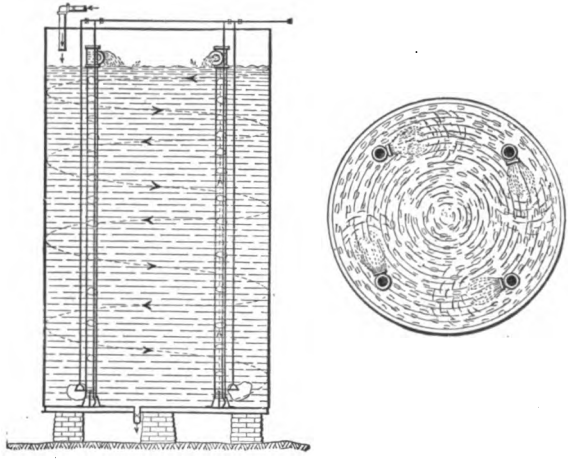

FIG. 81. Parral Agitator.

is lined with a porous inert material called 'silica-sponge.' Compressed-air is forced beneath this silica-sponge bottom, and rising through the interstices in the porous brick, ascends in innumerable small bubbles through the charge in the tank. Good results are reported to have followed its use in some Mexican mills, but it is hard to see that it has any advantages over the Parral or Pachuca agitator.

A. Grothe (*loc. cit.*) makes the following interesting remarks:

"It is sometimes adduced that the dissolving action of the cyanide is increased and accelerated by the quantity of air mixed with the solution, and the alleged rapidity of action in tanks with porous-brick bottoms is supposed to be due to the large quantity of air passing through the pulp in fine jets. A little reflection, however, will show the fallacy of this idea. In the Pachuca tanks the air enters the lift in a thin annular jet and mixes intimately with the pulp. There is no question of large bubbles when the tank is in uniform agitation, these only occurring at the time when it is necessary to employ more air for starting up. The air is carried downward for some distance with the falling pulp and rises in small bubbles to the surface. The solution in the tank is saturated with air, and it is clear

that the chemical action in certain kinds of ore can only be exerted by such air as is in solution, and not by the small bubbles on which stress has been laid. After the saturation point has been reached, any further admixture of air is ineffective, and this reasoning is fully borne out by experiment. I have repeatedly tried the effect of ultra-violent agitation as against that of a moderate one, and have found that as long as all particles are kept in motion, no further improvement either in point of time or ultimate recovery, can be looked for."

Intermittent vs. Continuous Agitation. Until quite recently all agitators were operated on the intermittent or 'charge' system. In this way, a tank was filled to its capacity with slime; this was agitated for the requisite time and discharged to the filter or filter storage. It then stood idle while being filled.

At the mill of the Esperanza Mining Co., at El Oro, Mexico, using 15 by 45 ft. Pachuca tanks, M. H. Kuryla * found that over 15 per cent of the net time of agitation was lost in filling and discharging the tanks. In addition much inconvenience and considerable cost was involved in discharging the tanks from the bottom, which was nearly 40 ft. below the top of the tanks used for pulp-storage prior to filtration.

To obviate these difficulties Mr. Kuryla connected the six tanks of the battery under consideration, by means of 6-in. pipe, feeding the pulp-supply to No. 1 tank, and withdrawing it from below the level of the top of the air-lift in No. 6 tank.

This method not only solved the problems mentioned, but showed a slightly increased extraction of both gold and silver, due probably to the increased time of agitation, incident to eliminating the filling and emptying of each tank. The evident advantages of this continuous method of agitation have led to its adoption in many of the large plants. It has been successfully applied not only to Pachucas but to Parral tanks, Trent agitators, and to other forms of agitating devices.

Time of Treatment. The time of treatment, whether by leaching or by agitation is, of course, a function of the individual ore, a derivative as it were of the ore's personal equation, but it will usually be found that a much shorter time of agitation with air is required, than where one of the mechanical or pump-agitators is used. It is also equally true that with the same ore far less time is required where the ore is agitated with solution, than where it is

* Mex. Min. Jour., August, 1910, Vol. 11, No. 2, pages 44–47.

treated by leaching alone. If our electrolytic solution theory is correct the reason for this is obvious enough, for with an adequate air-supply the likelihood of polarization is much lessened.

Effect of Extra-Solutions * and Oxidizers. For ordinary gold ores, air, provided it be furnished in sufficient quantity, seems to furnish oxygen enough for Elsner's reaction. But with telluride ores, cyaniding alone, without previous roasting, does not yield a satisfactory extraction. This problem has been solved in West Australia by the addition (in the agitator) of bromo-cyanogen to the cyanide solution after it has been in contact with the ore for 12 to 16 hours. From one-fourth to one-third the weight of cyanide is the amount of 'Bromo' used. The bromo-cyanide is prepared either by adding liquid bromine to cyanide solution packed in ice, or by agitating for a considerable period a solution of 'mixed salts' (potassium bromide and bromate) with potassium cyanide and sulphuric acid.†

For various reasons this treatment has not, until recently, been followed with the telluride ores of Cripple Creek, Colorado. The new Portland mill at that place is however employing a non-roasting process with success, the exact details of which have not been made public. There is reason for believing that the method is dependent upon the use of a halogen cyanide. Bromo-cyanogen is also used sparingly in the new Independence mill in the same district.

The Clancy process,‡ — patented — designed primarily for the treatment of these ores, employs cyanogen iodide as its active reagent. A solution of potassium iodide, cyanide, and sulphocyanide with calcium cyanamide is used in place of the usual cyanide. The ore is agitated with this solution while a weak electric current is passed continuously through the pulp, with the result that cyanogen-iodide is liberated. (Clancy suggests also the formation of cyanamidogen iodide (I_2CN_2).) Regeneration is effected by further electrolysis after addition of caustic soda to increase alkalinity.

* The decadence of hyposulphite lixiviation leaves this convenient term open to appropriation.

† E. W. Nardin, Min. & Sci. Press, Vol. 97, page 562.

‡ John Collins Clancy, paper read before the American Electro-chemical Society, Dec. 16, 1910, reproduced in Eng. & Min. Jour., Vol. 90, page 1250; Min. & Sci. Press, Vol. 101, page 862; Pac. Min., Vol. 18, pages 43–117. This is an exhaustive discussion and should be read by all interested.

With silver ores the problem is a different one; in certain complex ores — particularly when high-grade — there is some question as to whether air is a sufficiently energetic oxidizer on account of its low oxygen content. D. Mosher * has suggested ozone for this purpose, and reports good results from its use. Oxygen derived from electrolysis of water † has also been suggested for the same purpose. Other oxidizers and chemicals applied to the treatment of silver ores will be discussed elsewhere in this work.

* Pac. Min., Vol. 15, page 89.

† H. A. Megraw, Eng. & Min. Jour., Vol. 88, page 645.

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CHAPTER XI

SEPARATING SOLUTION AND PULP

So far we have been dealing mainly with the means for causing or aiding the solution of gold and silver in cyanide, and however completely we may be able to effect this vitally necessary operation, there still remains the problem — which in some cases is the most serious of all — of separating the valuable metal-bearing or 'pregnant' solution from the now worthless gangue matter.

Broadly, the methods for accomplishing this may be divided into four classes:

For sand { *I. Gravity filtration.* / *II. Gravity filtration assisted by pressure or suction.*

For slime { *III. Decantation.* / *IV. Forced Filtration.*

I. Gravity filtration is inseparably connected with the leaching method and has been already described under that process. The porous false-bottom of the tank serves as a medium through which the solution can flow and by which the sand is kept back. This is usually called 'natural filtration.'

II. Assisted gravity filtration is merely an extension of the foregoing principle. To the upper surface of the charge, or to the lower surface of the filter, pressure or suction respectively is applied, and there results, in some cases, a somewhat quicker and more complete separation of solution and tailing. The use of assisted filtration has been limited, however, by the tendency of the charge to pack under the extra pressure, with the consequent development of channels or fissures through which the solution would reach the filter without coming into uniform contact with the ore to be treated. On the whole, assisted natural filtration has been a thorough success in only a few cases. Recently, good reports have been received of its use in South African practice.*

III. Decantation, following a period of undisturbed settlement, was for many years the only way known for the separation of

* W. A. Caldecott, Min. & Sci. Press, Vol. 98, page 316.

cyanide solution from slime. It had, of course, the inherent defects of any system based upon the principles of Mark Twain's Calaveras frog jumping half his preceding distance at each successive attempt. Nevertheless, in South Africa, at least, decantation had an immense vogue. Its popularity, however, like that of the stone adze or the flint arrow-head, was a matter of 'nothing better known.' For small or temporary equipments, intermittent settlement and decantation may occasionally be desirable, but in

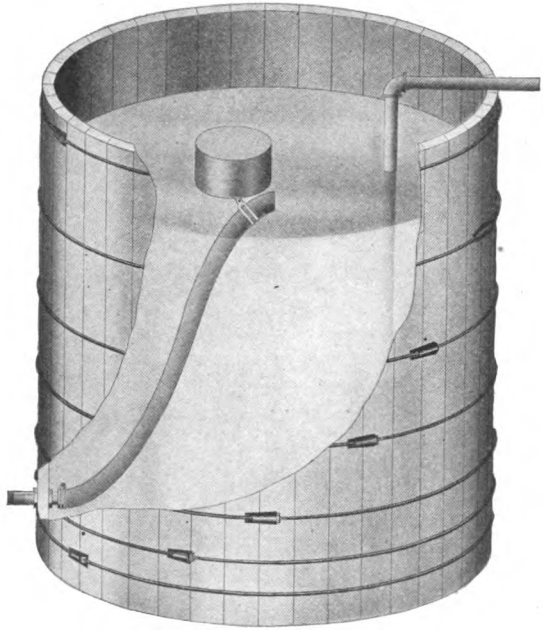


FIG. 82. Decantation with Hose and Float.

the main the process in its original form has passed into cyanide history. A few facts as to the details are of interest however.

The settlement usually takes place in deep tanks, as it is generally believed that the deeper the column of settled slime the dryer will the lower strata become, although some recent investigations would seem to prove otherwise.* It is usually estimated that slime retains its own weight of solution after agitation and settlement, so that one can easily approximate the value left after each washing. The amount of wash water required depends upon the

* R. T. Mishler, Eng. & Min. Jour., Vol. 94, page 643 (1912).

value remaining in the tank after each dilution and incidentally upon the volume of solution it is possible to decant. Various reagents to promote flocculation and settlement of the slime have been used with success. Lime is the most commonly employed substance for this purpose. In crude equipments, the side of the tank is perforated with a vertical series of holes, so arranged by cocks and spigots that the outlet may be at any desired point below the surface. Siphons with floats to keep them drawing from the immediate vicinity of the surface may be employed, and in the South African plants a float attachment to a hose or pipe with a loose elbow (Fig. 82) near the bottom of the tank has been successfully employed.

With the perfecting of appliances for the de-watering or thickening of pulp, there have been many proposals of 'continuous-

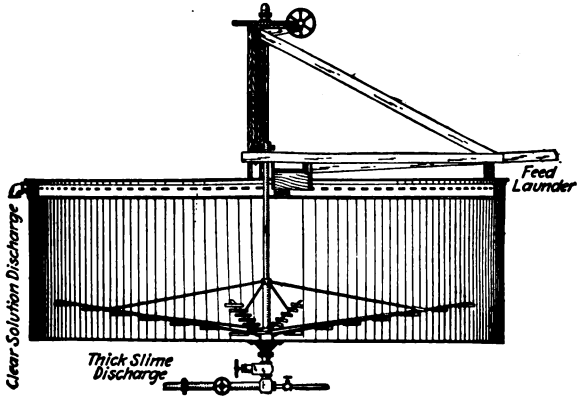


FIG. 83. Dorr Continuous Thickener.

settlement,' or 'continuous decantation,' as a means for the separation of pulp and pregnant solution.

Of these proposals that made by J. V. N. Dorr, elsewhere mentioned, and embodying the application of the Dorr continuous thickener to counter-current decantation, seems to be meeting with most success. This thickener, Fig. 83, consists of a flat-bottom tank with a peripheral-launder overflow, and a central spigot discharge, fed by the gentle rabbling action of radial arms mechanically operated.

The method, as employed in supplementing the usual filtering equipment, is shown in Fig. 84, from which it will be seen that

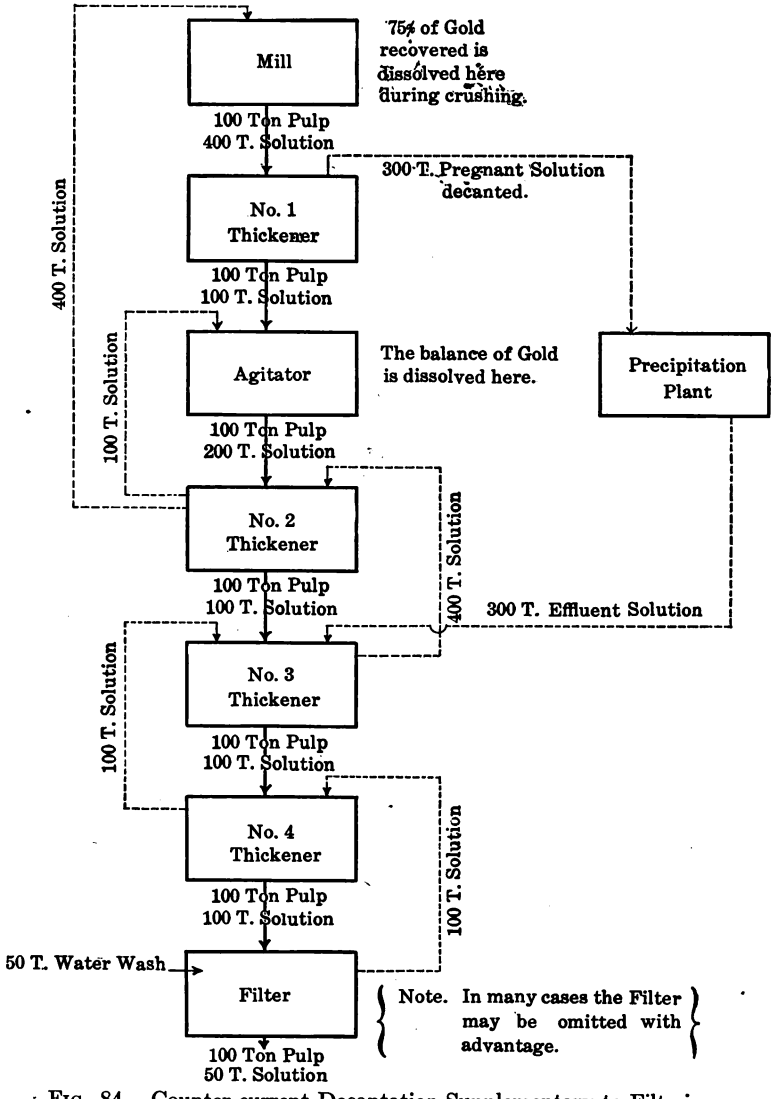


FIG. 84. Counter-current Decantation Supplementary to Filtering.

after the first thickener, the flow of pulp continually being impoverished is in the opposite direction to the flow of solution continually being enriched. Finally, the solution, having accompanied the ore through the crushing stage, is decanted from the first thickener and goes, after clarification, to the precipitation plant. The number of thickeners employed is, of course, a function of the strength and richness of the solution and the character of the pulp.

In certain cases, *e.g.*, where weak solutions can be used, and where the pulp is of such character that it can be discharged from the thickeners with a low moisture content, it appears not only possible but decidedly profitable to employ continuous decantation without a supplementary filter. In the cyaniding of silver ores where strong solutions are necessary, or in the treatment of an ore carrying heavy sulphides, which would have a tendency to settle and be discharged from the system before a sufficient time for treatment had elapsed, the method would hardly be applicable.

These limitations, it is only fair to say, are recognized and insisted upon by Mr. Dorr himself.

IV. Forced filtration. When we come to forced filtration we are confronted by an almost endless variety of apparatus, and it is very certain that the end is not yet. Two distinctive types are in use, which may be designated as filters of the press type and filters of the vacuum or suction type.

(A) **Filter-Presses.** The filter-press, as first used in sugar-making and in the chemical industries, was for many years used in cyaniding, practically without change in the design. Essentially, it consists of a number of flat cells or chambers with walls of canvas or other filtering medium. The pulp is charged into the chamber, either from a gravity-system, or from a montejus or storage-tank so arranged that the pulp is displaced by air-pressure. The liquid passes out through the pores of the cloth into suitable channels, while the solid material remains between the cloths in the form of a cake. Provision is made for washing the cake thus formed and for displacement of the wash solution by compressed air. For details of the principles of construction reference should be made to Figs. 85 and 86 of the Merrill press, keeping in mind the fact that, aside from the manner of discharging, the two types are identical. The press is emptied by releasing the hydraulic or screw-gear keeping the different units together, the frames are slid apart, and the cake is removed by hand. The capacity of a press

of this type with 50 frames, each 40 in. square, is usually from 4 to 5 tons of dry slime per 24 hours. The cost of operating is from 50 cents to $1.00 per ton, and is usually nearer the higher figure. The obvious objection to the use of hand-removal presses is their high cost, both for installation and operation. Nevertheless, they have been extensively used in Australia and South Africa. In North America, however, they never became popular, probably because of the high price of labor which would make their operation almost prohibitive.

The necessity of solving the problem of treating the large amount of slime produced by the Homestake mills, which assays less than a dollar a ton, has led to the development of a filter-press which does away entirely with the hand-removal feature.

The Merrill filter-press as described by Clarence G. Semple,*

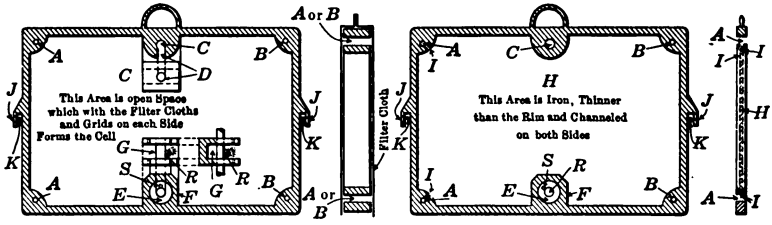
"consists of an iron frame in which are placed 90 cell- or chamber-frames, with distance-frames or grids between them, to control the direction of the flow of solution through the cake of slime.

A diagram of a cell-frame is shown in Fig. 85. It is a rectangular open casting with rounded corners, 4 in. wide, with the edges machine finished. In the four corners are two 2½-in. holes *AA* and *BB*. In the center of the upper part of the frame there is a 4-in. horizontal hole *C*, from which a passage *D* opens downward into the chamber or cell. At the bottom of the frame there is a 6-in. hole *E*, passing through the lug-like projection *F*, which projection is open through the center, as shown at *G*, opening into the cell upward and to both sides. The grid or distance frame is shown in Fig. 85. It is exactly the same size as the cell frame, but only 1 in. thick. The edges of the plate are machine finished. The entire frame is a solid casting; the area within the rim *H* is a little thinner than at the rim and is grooved or fluted on both sides, like the surface of a laundress' washboard, only the grooves do not all run in the same direction. There are four 2½-in. holes *AA* and *BB* in the corners in exact juxtaposition with the corner holes of the cell frame, but the *AA* holes have a small channel *I* leading from the holes to each side of the fluted surface *H*. The grids have the holes *C* and *E* just like the cell frames, but are continuous through the casting without any lateral passages or channels.

"The filter-cloths are cut large enough to cover both sides and hang over the top of the cell frames, as shown in Fig. 85. Discs are cut out of the cloths opposite each of the six horizontal holes in the casting. Cotton-twill cloth is used covered by 8- or 10-oz. duck. All the frames have lugs *J* cast on the sides with rollers *K* upon which the frames rest when

* Eng. & Min. Jour., Vol. 91, pages 719–722.

placed in the press. The method of making up a press is shown in Fig. 86. A grid frame *M* is placed against the head *L* of the press with a gasket between, in which six circular holes have been cut in line with the horizontal holes in the grid; P. & B. paint is used on the edges to make a tight joint. A cell frame *N*, with the filter cloths in position, is then set tightly against



Cell Frame. Distance Frame.

FIG. 85. Merrill Press, Plate and Frame.

the grid and a second grid *O* placed against it, but in the reverse position to the first grid so the corner holes with the channels *I* are on the opposite side of the press than the corresponding holes in the first grid. The second cell-frame is then placed and the third grid brought up against it, but with the channels *I* on the same side as the first grid. The press is thus filled with frames, with every alternate grid with the channels on the same side of the press, until the 90 frames are in position. The frames are then drawn tightly together by draw bolts operated by capstan heads. The six horizontal holes through the frames now form six continuous passages or tubes through the press that are blind against the back head *P*, but which pass through the front head *L* and connect with pipes outside the press.

"**The Operation of the Presses.** The passage *C*, through the upper part of the press, is connected with the pipe from the slime-storage tank. By opening a valve on this pipe the slime enters the press through *C*, and passes into the interior of the cells through the passages *D*. When the cells have been filled, the supply valve is closed, and air at 26 pounds pressure is admitted through the passages *AA*, which flows through the press, as shown in Fig. 86, coming out from the opposite side of the press through the passages *BB*. The air removes the excess of water and aerates the slime cake. The first period of aeration takes about two hours, and is succeeded by the following operations: Treatment with strong solution 0.12 to 0.14 KCN; draining strong solution; aeration; second treatment with strong solution; draining; treatment with weak solution 0.07 to 0.075 KCN; draining; aeration; second treatment with weak solution; draining; treatment with wash water; and discharging the press. The cycle from filling to discharging takes about nine hours, the treatments

with solution being a little longer than the periods of aeration, except the first blowing.

"The arrangement of the piping outside the press is shown in Fig. 86. By this arrangement of the valves, air, water, strong or weak solution, may be admitted from either side of the press and drawn out from the side opposite. The usual practice is to admit the solution at one side of the press for one-half the period of treatment, and then to reverse the flow by the rearrangement of the valves. The effluent solutions and wash water discharge into a small iron sump at the front end of the press.

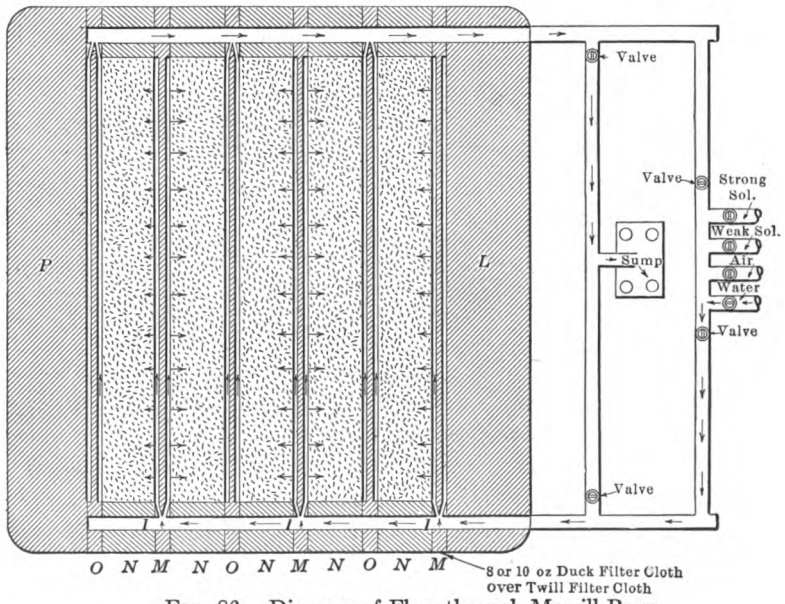


FIG. 86. Diagram of Flow through Merrill Press.

"**Discharging the Slime Cake.** The slime cake is discharged through the bottom central passage. Through the passage *E* there runs a 2-in., or 2½-in. pipe *R*, Fig. 85. This pipe enters the front header through a stuffing-box and enters a bearing socket in the back header of the press, and is supported outside the press, by boxes, just as a shaft would be. In each cell-compartment there is a little nozzle *S* screwed into this pipe. Four of the cell-frames, at equal intervals along the length of the press, are tapped at the bottom to receive a short piece of 2½-in. pipe that can be closed by valves. When the press is to be discharged these valves are opened and water under pressure enters the press through the pipe *R*, which is turned back and forth through an angle of 170° by mechanism outside the press. A jet of water enters the cells through the noz-

zles *S*, loosening the cake of slime that runs through the annular passage left between the outside of the pipe *R* and passage *E*, until reaching one of the cells with the discharge pipe, through which it flows into the launder leading to the creek."

The presses are 46 ft. long and weigh 65 tons each; the frames measure 4 by 6 ft. outside. On the Homestake slime their capacity is 25 tons per charge. The recovery is about 90 per cent or 81 cents per ton at a total cost of about 25 cents, of which the actual filtering cost is less than 11 cents per ton.

As used at the Homestake all the treatment is given in the presses. Where, as in the case with certain ores, a longer time of treatment is necessary, solution may be effected in the ordinary agitating-tanks, and the Merrill presses used merely for the separation of solution and pulp.

Another modification * in the employment of the Merrill presses has been to avoid the formation of a solid 4-in. cake, the slime-feed being shut off, while there is a space of $\frac{1}{4}$ in. in the middle of the frame. Water or solution is then introduced through the sluicing pipe, leaching taking place from the center outward. The general opinion seems to be that the Merrill press is highly successful in handling finely-crushed ore free from colloidal material, but that where the ore forms a tough clinging slime, difficulty is experienced in completely sluicing out the cake.

(B) Vacuum-Filters. In cyaniding, the finer the product the higher the extraction; this is the case with few exceptions. To apply this principle requires two things: a satisfactory machine for fine-grinding and a filtering scheme which would be at once economical and efficient. Filter-pressing, while an established practice in Australia, was never popular in America and few successful installations are recorded prior to 1907, the date of introduction of the Merrill press.

About 1903 George Moore of the Consolidated Mercur Company, in Utah, devised a vacuum-filter and installed a plant at the Mercur mill. This, while faulty in design and a comparative failure so far as cyaniding is concerned, was the origin of the vacuum type of filter, recent modifications of which are in use at a number of mills in this country and in Australia, and are being introduced into South Africa.

The unit of the *Moore filter*, as now used, is a rectangular frame

* Henry Hanson, Min. & Sci. Press, Vol. 98, page 657.

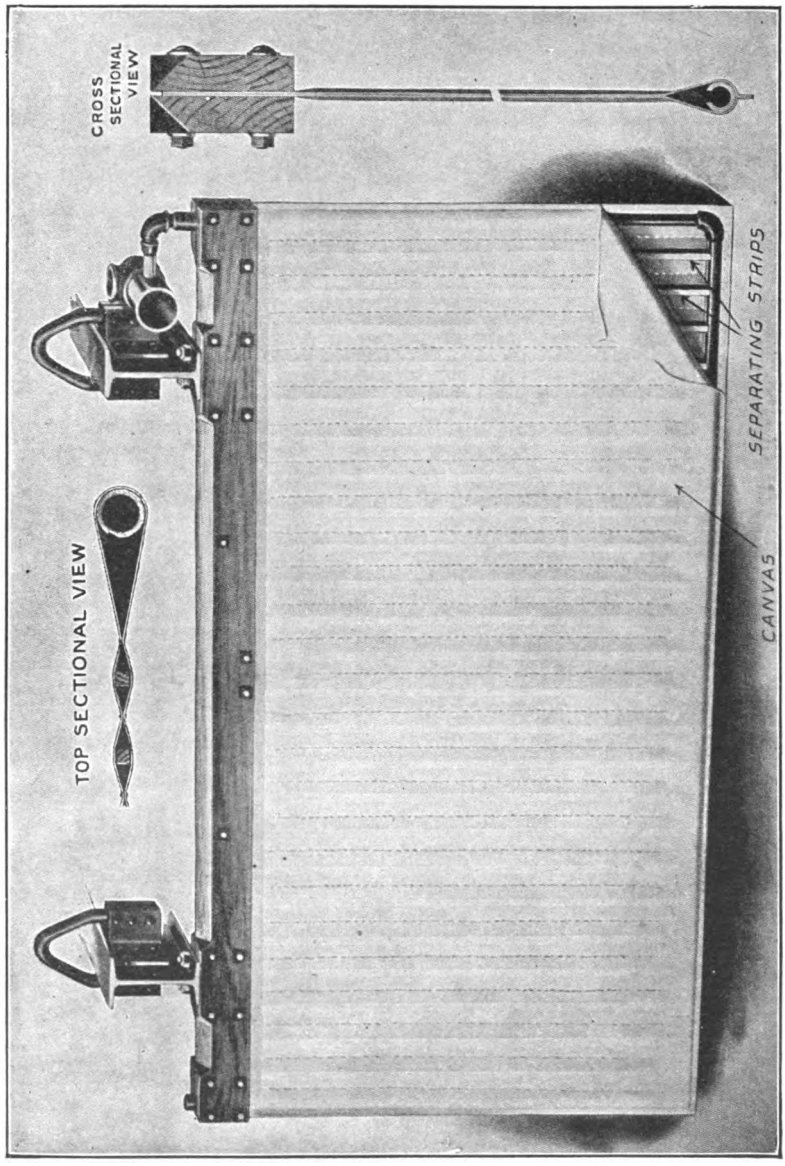


Fig. 87. Moore Filter Leaf.

(Fig. 87) 5 by 10 ft., made of ½-in. pipe,* with a wooden header. The lower member of the pipe frame has its upper surface perforated with ¼-in. holes to admit the filtered solution. The filter consists of a sheet of canvas stretched over the entire pipe frame on both sides. Vertical wooden strips, spaced 4 to 5 in. apart, are used as stiffeners and to keep the filter from collapsing. Between the stiffeners the two sides of the filter are sewed tightly together.

In operation these frames or 'filter-leaves,' as they are called, are grouped in clusters or baskets, and in the original Moore process, several large installations of which are in operation, the basket

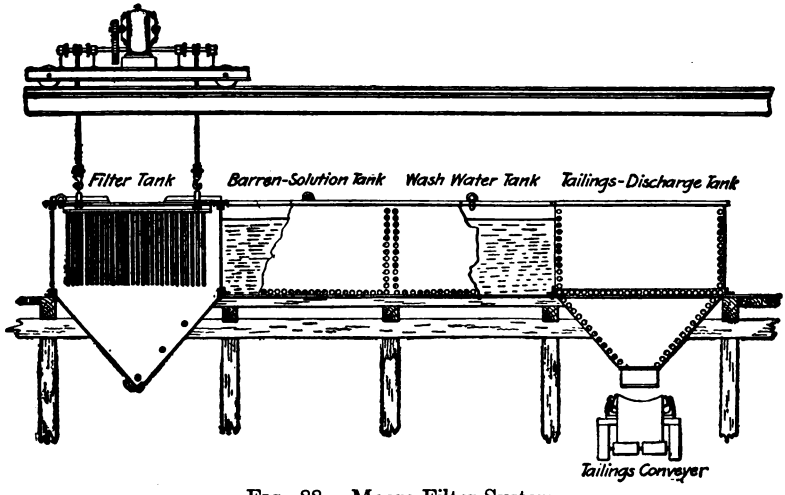


FIG. 88. Moore Filter System.

of frames is lowered by suitable means into a hopper-bottom compartment containing the slime pulp (Fig. 88). The vacuum is applied to a common pipe connected with each frame, the solution is drawn through the canvas by means of a vacuum pump, and a slime cake, varying from ⅞ to 2 in. in thickness, is deposited on each side of the filter-frame. The cluster of filter-frames, carrying the charge of slime and weighing several tons, is then lifted from the pulp, and is shifted automatically to adjoining compartments containing the wash of weak solution and water respectively. Into

* Moore's early filter did not have the pipe frame, for which idea, apparently, cyaniding is indebted to G. A. Duncan, see Min. & Sci. Press, Vol. 96, pages 769–770.

each of these it is lowered in turn, the vacuum applied, and the displacing operation carried on. The load is finally raised and shifted to a bin, where the cakes are discharged by introducing air or water into the interior of the frames.

The vacuum during the transfer of the cake calls for careful regulation; if too high, the cake is likely to crack; if too low, the cake may become detached from the filter surface.

It was, of course, not a difficult matter to modify this idea to the extent of keeping the filter stationary, bringing pulp, weak solution and wash-water, into the filter compartment in turn and then dis-

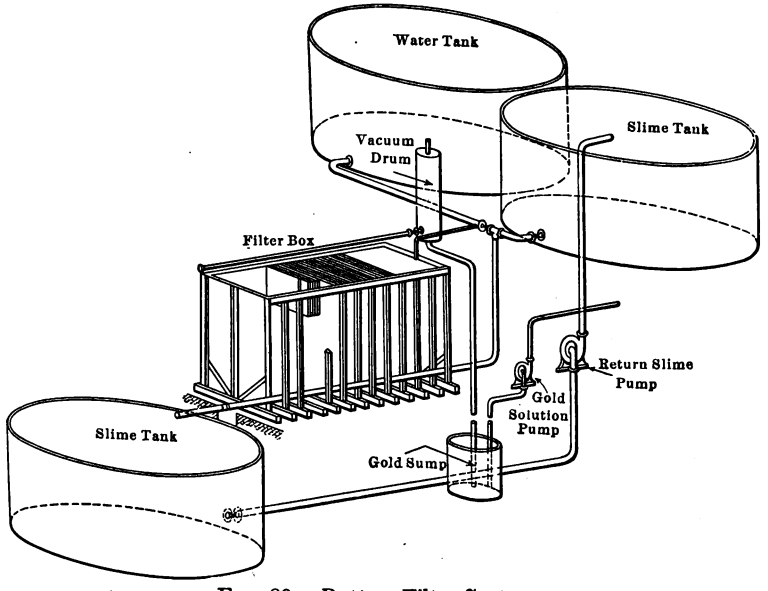


FIG. 89. Butters Filter System.

charging the spent cakes into the same tank. This, with some mechanical improvements, constitutes the Butters' modification of the Moore filter.

In the *Butters filter* (Fig. 89) the leaves are set in a rectangular box or tank, the bottom of the box consisting of a series of pointed pockets to facilitate the discharge of the spent cakes. The frame is similar to the Moore, the filter, however, consisting of a piece of cocoa-matting, fitting inside the pipe-frame, upon both sides of which canvas is quilted by frequent sewing; stiffening ribs of wood are placed vertically outside the filter every foot or so apart. The

frames stand parallel in the filter box at about $4\frac{3}{4}$ in. centers. The slime pulp is drawn from the slime reservoir or pumped, if necessary, into the bottom of the filter box until all the frames are immersed. The vacuum is then applied until a cake of suitable thickness is deposited; and the excess of pulp is returned to the slime reservoir. This operation is repeated for the wash, and the cake is finally discharged into the bottom of the box by introducing water under a low head into the interior of the leaves. The accumulated cakes from each discharge are removed by sluicing.

In cost of operation the two types are about equal, running from 5 to 22 cents per ton of dry slime. This variation in cost is largely a matter of the character of the slime, which influences the time required to form a cake of suitable thickness, as well as the amount of washing required to displace the cyanide solution. The average cycle of treatment, including charging, caking, washing, and discharging the cake, varies from $2\frac{1}{2}$ to 5 hours. On an average about 50 to 75 sq. ft. of filtering surface per ton of slime capacity per day is necessary. The Barry and Banks system, in use in New Zealand, and known as the Crush filter, is to all intents and purposes the same thing as the Moore.

These 'immersion-filters,' as they have been termed, marked a most distinct advance in the cyaniding art, and in conjunction with the tube-mill made possible results hitherto unattainable. In common, however, with the Merrill press, with which at the present time they share the greater part of the world's cyanide tonnage, they are open to the objection of being discontinuous in operation, that is, they work intermittently, thus failing to coordinate nicely with the rest of the plant, where crushing, amalgamation (if it is practiced), concentration, and even agitation in many plants are continuous processes. To obviate this objection, various vacuum-filters of a traveling or continuous type have been proposed.

The Ridgeway filter, shown in Fig. 90, is one of the earliest of this type. This machine consists of a number of cast-iron filter-plates, sectors of an annulus in shape, which revolve in a horizontal plane about a hollow central spindle. Over the underside of these filter-plates, cloth or canvas is stretched, the space between the canvas and the frame forming a chamber which has suitable connections with vacuum pump and with air-pressure at different parts of the cycle. In operation, as it revolves about the central spindle, each

plate dips in turn into the part of the annular trough containing slime pulp, accumulates a cake on its surface, rises out of the slime pulp and dips into wash water which washes the cake; the plate

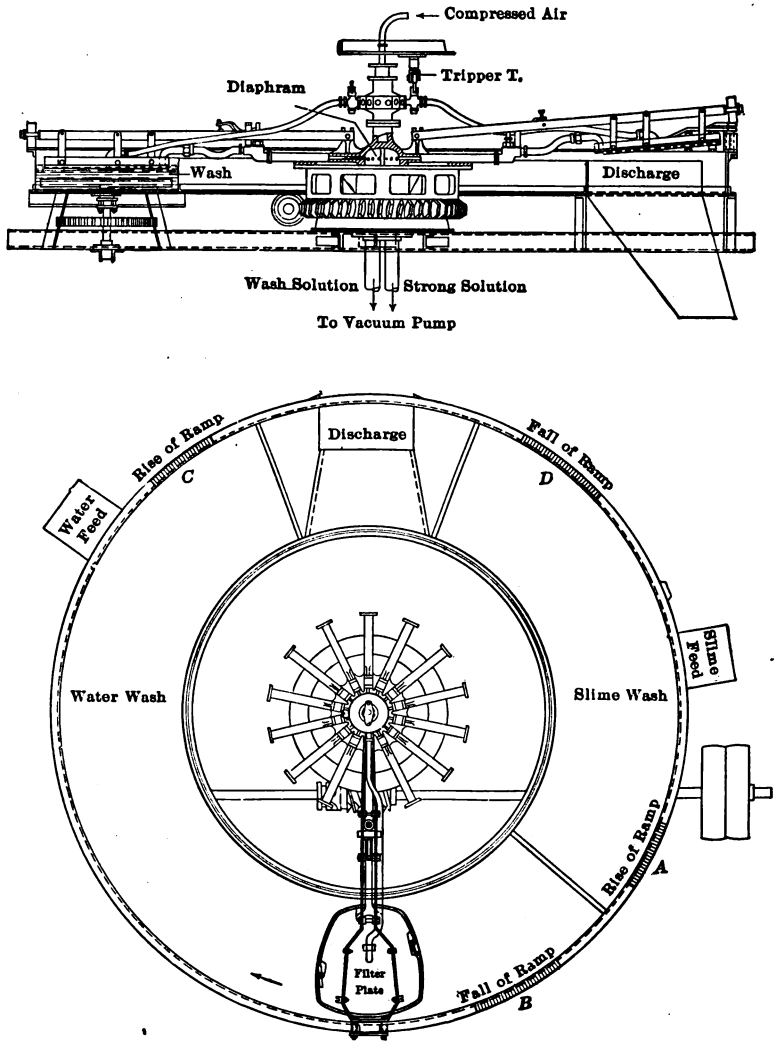


FIG. 90. Ridgeway Filter.

then rises and descends again in its circular path to the discharge point, where, by a suitable mechanism, the vacuum-connection is shut off, and air-pressure is applied, thus releasing the cake. The

particular sector we have been following is then ready to begin the cycle over again.

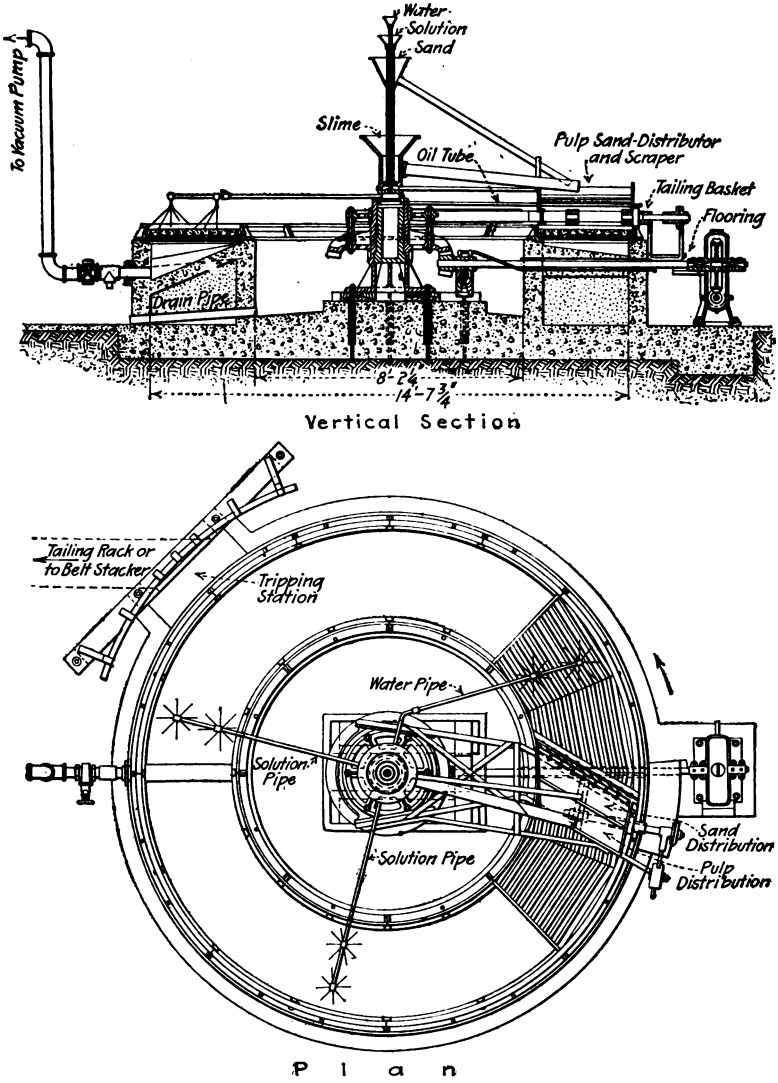


FIG. 91. Hunt Filter.

The Hunt filter (Fig. 91), as described by the inventor,* consists of a horizontal, annular chamber which is connected with a

* Bertram Hunt, Min. & Sci. Press, Vol. 97, page 430.

vacuum-pump, the top of the annular chamber being built on the principle of the sand-filter, described under 'Leaching-Tanks.'

"This chamber forms the permanent filter-bed and is supported strongly enough to remain perfectly rigid on the application of a vacuum in the chamber underneath. This chamber is connected with a wet vacuum-pump which withdraws the solution and maintains the vacuum.

"The annular vacuum-chamber is made of concrete and iron and the walls, being raised above the top of the filter-bed, constitute circular tracks on which a carriage revolves. This carriage has an iron scraper in front, which removes the layer of residue from the permanent filter-bed. Fol-

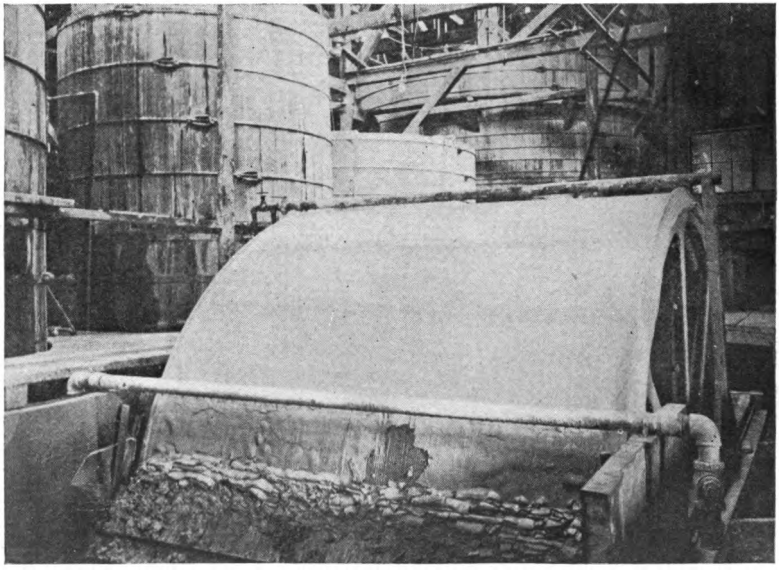


FIG. 92. Oliver Continuous Filter.

lowing close behind the scraper, on the carriage, is placed a distributor similar to that used on a Frue vanner, which handles the sandy portion of the pulp. Behind this is a second distributor which spreads the slimy portion of the pulp over the layer of sand just laid down by the first distributor. On the central column of the machine are two concentric hoppers or funnels; these are connected by pipes with the two distributors which receive the pulp. Before reaching the filter machine the pulp is roughly classified, so that a portion of the fine sand is separated in a clean condition, and this clean sand is fed upon the filter-bed by the first distributor, the rest of the pulp going to the second. In this way the slimy portion of the pulp is always distributed over a layer of clean fine sand.

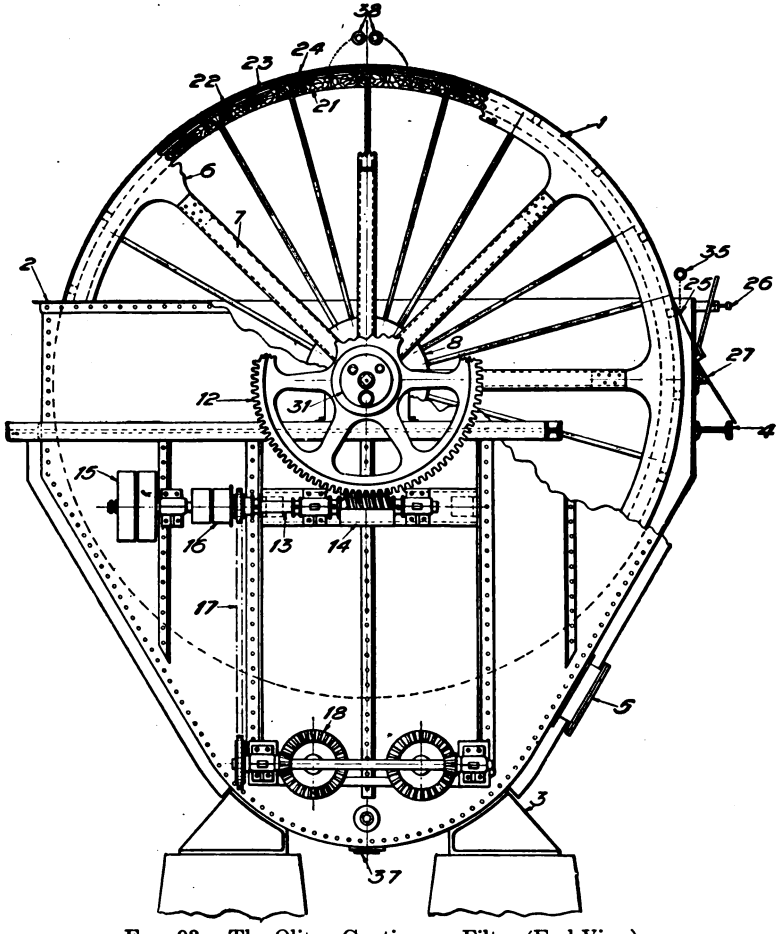


FIG. 93. The Oliver Continuous Filter (End View).

List of Parts.

1. Filter Drum.
2. Steel Filter Tank.
3. Cast Iron Pedestals.
4. Steel I-Beam Frame.
5. Manhole.
6. Cast-iron Spider Rim.
7. Channel Steel Arms.
8. Hollow Trunnion.
9. Steel Shaft.
10. Main Bearings.
11. Stuffing Boxes.
12. Worm Drive Gear.
13. Worm Shaft.
14. Oil Well for Worm.
15. Filter Drive Pulleys.
16. Pulleys for Agitator and Wiring.
17. Chain Drive for Agitator.
18. Bevel Gears on Agitator Shafts.
19. Agitator Shafts.

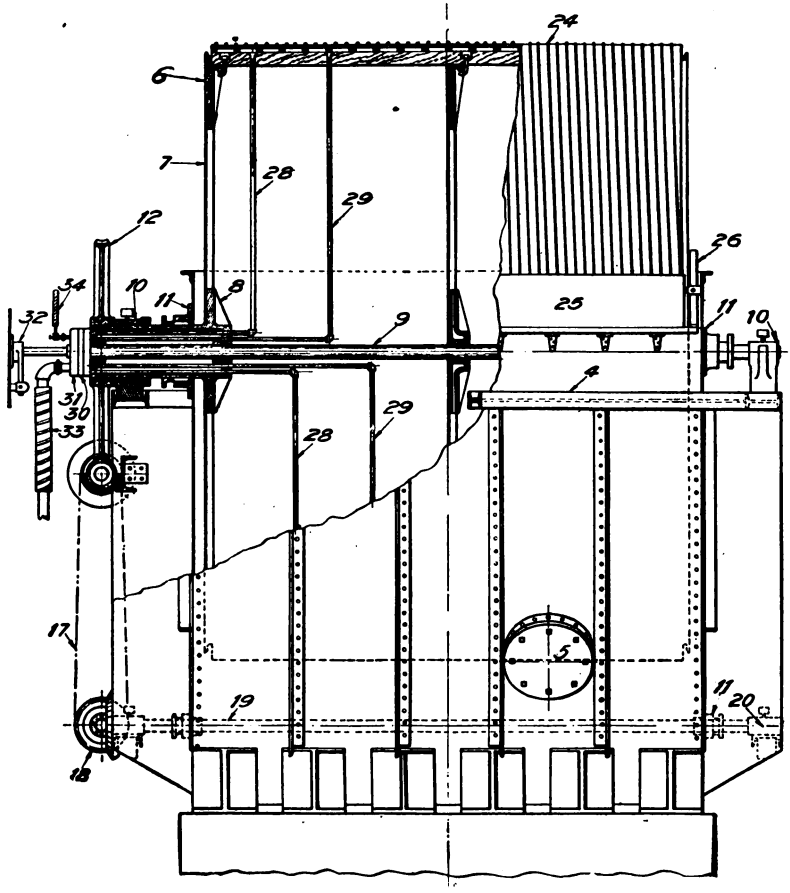


FIG. 93A. The Oliver Continuous Filter (Side View).

List of Parts.

20. Agitator Shaft Bearings.
21. Wood Staves for Drum.
22. Section Division Strips.
23. Filter Medium.
24. Wire Winding.
25. Steel Scraper.
26. Scraper Adjustment.
27. Tailing Apron.
28. Vacuum Pipes.
29. Compressed Air Pipes.
30. Regrinding Valve Seat.
31. Automatic Valve.
32. Adjusting Lever for Valve.
33. Vacuum Hose Connection.
34. Compressed Air Connection.
35. Discharge Spray Pipe.
36. Emergency Agitator Pipe.
37. Drain Flange.
38. Wash Water Pipes.

As the carriage moves continuously the deposited layer of sand and slime is also continuously removed by the scraper, while fresh material is deposited by the distributors, as described above. Pipes to spray wash-solution and wash-water follow the carriage at suitable distances, so that the residue can be thoroughly washed."

Another device quite similar in general design to the Hunt is the *Parrish filter.** In this, however, the filter surface is of canvas or cocoa-matting protected by wire winding.

The Oliver filter (Figs. 92, 93 and 93A). Probably the most successful of all the continuous filters is that perfected by E. L. Oliver at the North Star mine, Grass Valley, California. This machine consists of a drum or cylinder with open ends, revolving about a horizontal axis and dipping into a tank containing (under gentle agitation) the pulp to be filtered. Usually about half the drum is submerged. The outer surface of the cylinder forms an annular vacuum-chamber, divided into sections and covered with filter cloth protected from wear by a wire winding.

By suitable valve mechanism at the end of the trunnion, and by corresponding pipes communicating with the filter-sections, vacuum is applied to form the cake during the submergence of each section; this vacuum is continued, as the section rises out of the pulp, until it passes the wash-water. Finally the vacuum connection is cut off and air pressure applied just as the section reaches the discharge point; the cake thus loosened is removed by a light steel scraper and the filter-section continues in its revolution into the pulp to commence another cycle. The thickness of the cake varies, according to the character of the slime, from $\frac{1}{4}$ to 1 in. The time of one revolution is from 5 to 10 min. for the standard filter, which is 11 ft. 6 in. in diameter and of any length, up to 16 ft. From 3 to 5 sq. ft. of filter area is required per ton of daily capacity.

The Portland filter of the Colorado Iron Works is substantially of the same type.

As a type intermediate between filter-presses and vacuum-filters, we have the pressure-filters, which in general construction bear more resemblance to the submerged, vacuum type than to the presses. All of these offer the general advantage of allowing the use of a higher pressure than it is possible to obtain from the at-

* Pac. Min., Vol. 15, pages 124–125.

mosphere, and for this reason are particularly suited to use in high altitudes.

The Blaisdell filter (Fig. 94) is virtually of the Moore-Butters type, consisting of vertical filter-leaves suspended in a vertical pressure-cylinder.

The Kelly filter (Fig. 95) is similar in principle, except that the pressure-cylinder is set at a slight inclination to the horizontal, and the cylinder head, carrying the basket of filter-frames, moves up the slope into the cylinder on a suitable track. Pressure is applied, and the cake accumulated and washed as usual; the head with its accompanying filter-frames and cakes is allowed to run down the track until clear of the cylinder, when the cakes are discharged in the usual manner. A press of this type with a cylinder 4 by 9½ ft., and having 300 sq. ft. of filtering area, is stated to have a capacity on roasted telluride ore of 50 tons per day.

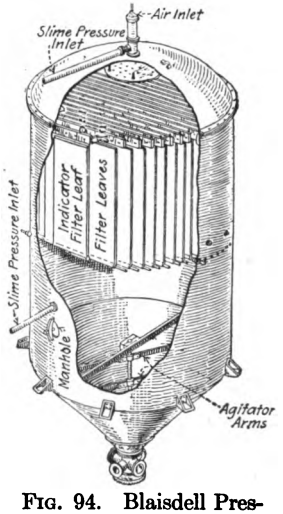


FIG. 94. Blaisdell Pressure-Filter.

The Burt pressure-filter (Fig. 96) consists of a pressure-cylinder inclined at 45°, with the head at the lower end operated by toggles. Within the cylinder are suspended a number of ordinary filter-frames, set transversely to the cylinder axis. The cylinder is filled with pulp under pressure, cake accumulated, excess pulp discharged, wash-water introduced, cake washed, and finally discharged by opening the discharge-door or lower head. At El Oro, Mexico, with cylinders containing 28 frames, each of 8 sq. ft. area, 120 tons of dry slime are filtered per 24 hours.

The Burt revolving filter is a recent innovation which looks decidedly promising. It consists of a revolving cylinder similar in appearance to a tube-mill, the filter surface forming the inside walls of the cylinder. Pulp is charged into the interior, air pressure turned on and the cylinder slowly revolved. The cake accumulates about the periphery, and the solution flows out through suitable holes in the shell to a sump beneath. When all the solution has been expelled and air blows from the holes, wash-water or weak solution is turned in, and the cake is washed.

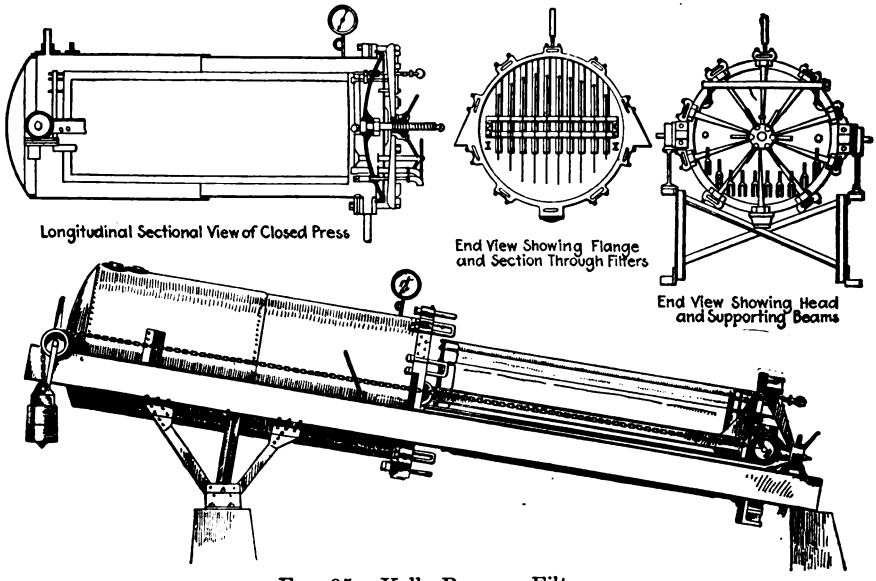

Longitudinal Sectional View of Closed Press
End View Showing Flange and Section Through Filters
End View Showing Head and Supporting Beams

FIG. 95. Kelly Pressure-Filter.

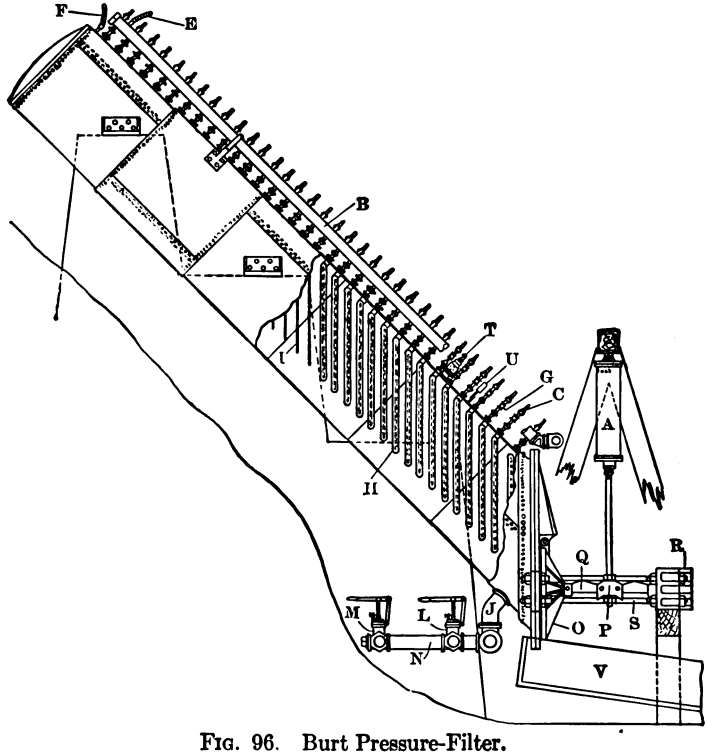


FIG. 96. Burt Pressure-Filter.

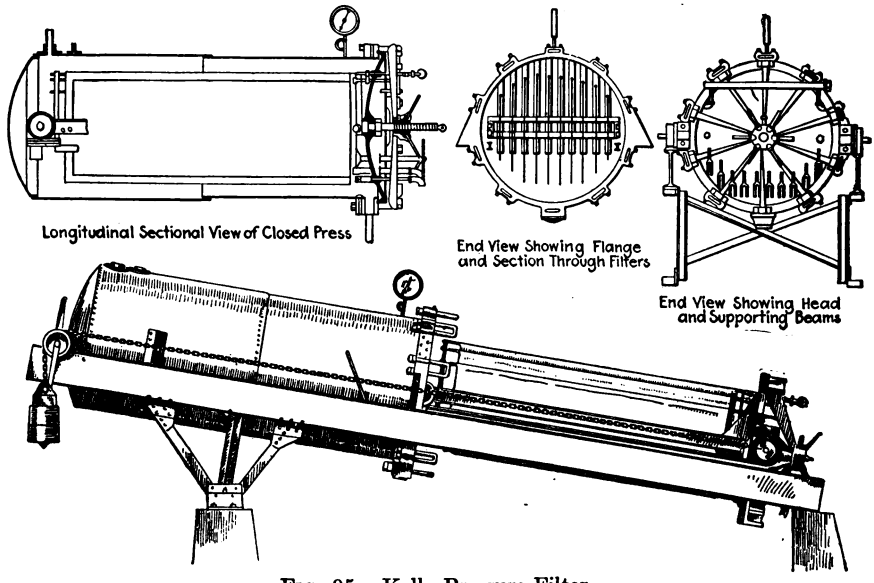

Longitudinal Sectional View of Closed Press
End View Showing Flange and Section Through Filters
End View Showing Head and Supporting Beams

FIG. 95. Kelly Pressure-Filter.

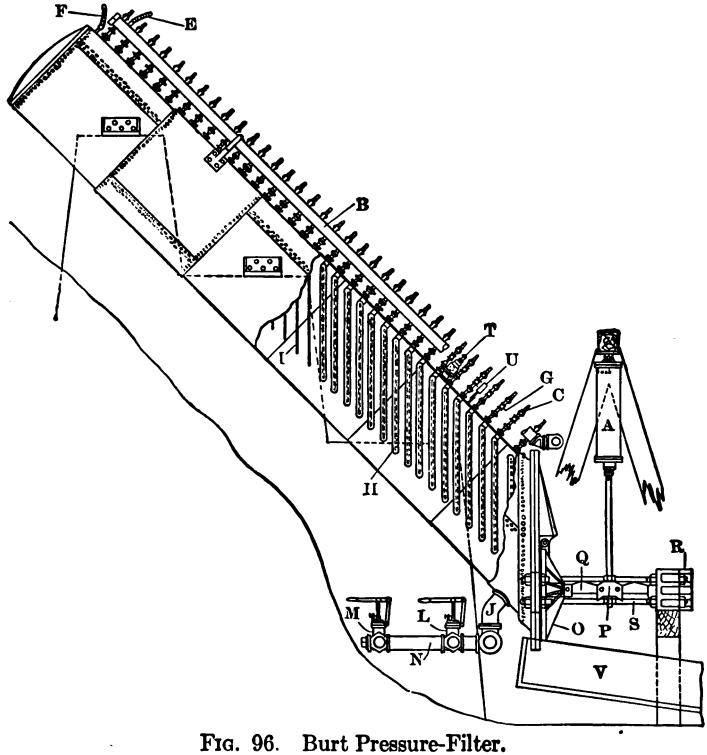
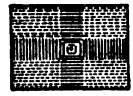


FIG. 96. Burt Pressure-Filter.

Name or type	Methods of inducing filtration	General shape, usual size, position, and material of filtering surface	Nature of filter support	Thickness of cake	Time of cycle	Method filter an juxta
Plate and frame press.	Pressure from pump or montejus.	Rectangular, 40 in., square vertical. 10- to 12-oz. duck.	Corrugated or studded C. I. plates.	2 to 3 in.	1½ to 5 hr.	Forced ter c
Merrill.	Gravity pressure.	Rectangular, 3×5 ft., vertical, 10- to 12-oz. duck over cotton twill.	Corrugated thus 	4 in.	9 hr., including treatment in presses.	Gravit stor
Merrill.	Gravity pressure.	Rectangular, 3× 5 ft., vertical, 10- to 12-oz. duck over cotton twill.		1¾ in. each side	4 hr.	Gravit stor
Moore..	Vacuum pump.	Rectangular, 5×10 ft., vertical, 10-oz. duck.	Pipe frame with slats, between which canvas sewed together	1 to 2 in.	2½ to 5 hr.	Transf char disc
Butters.	Vacuum pump.	Rectangular, 5×10 ft., to 10×10 ft., vertical, 10-oz. duck.	Pipe frame, with cocoa-matting support, to which canvas stitched, or on wire netting.	1 to 2 in.	2½ to 5 hr.	Pulp fron or st
Ridgeway..	Vacuum pump.	Annular sector, 3 sq. ft., horizontal, face downward, 10-oz. duck.	Cast-iron frame, canvas stretched over	¼ to ⅜ in.	1 min.	By tr sma
Hunt...	Vacuum pump.	Annular ring, horizontal, sand surface.	Triangular bars, apex upward, with gravel cover.	¼ to ½ in.	N. D.	Sand wasl tach
Oliver..	Vacuum pump.	Barrel 11 ft., 6-in. diam. × 7 to 16 ft., horizontal, 12-oz. duck, wire-wound.	Corrugated staves with burlap.	¼ to ½ in.	3 to 5 min.	Filter in ta full
Blaisdell...	Pressure.	Like Moore and Butters but smaller leaves in vertical cylinder.	Like Moore and Butters.	¾ to 1½ in.		Similar
Kelly...	Pressure.	Narrow, rectangular, vertical, canvas covered, leaves in nearly horizontal cylinder.	Pipe frame, cocoa-matting, or wire netting	1 to 3 in.	30 to 45 min.	Pulp c unde from gal p mont
Burt ... (Pressure).	Pressure.	Small leaves, transversely set in 45° cylinder.	Pipe frame, cocoa-matting, or wire netting	¾ to 1½ in.	30 to 45 min.	Pulp cl der p sure gravi
Burt ... (Revolving).	Pressure.	Lining walls of cylindrical chamber, 42 in. diam. ×40 ft. Filter in sections 26 in. × 10 ft. each.	Corrugated staves set in filter sections.	Up to 6 in.	60 to 90 min.	Pulp cl gravi tion comp air.
Sweetland..	Pressure.	Small leaves, 6×4 ft., in filter press frame, canvas surface.	Pipe frame, cocoa-matting, 15-oz. duck.	1 to 1½ in.	35 to 60 min.	Pulp ch into by m

* This cost is inclusive of pulp-transfer, cloth-renewals, acid-wash, etc. The ne

Bringing slime into action	Method of washing	Method of discharging	Reported costs per ton dry slime	Filter particularly adapted for handling	Advantages	Disadvantages
Frue filters.	Water forced into alternate filter spaces and thru cake.	Hand-removal.	25¢ to $1.00	Granular slime.	Good washing.	Cost of operation and installation.
Grain tanks.	Water forced into alternate filter spaces and thru cake.	Sluicing.	9½¢*	Granular slime.	Great capacity, low cost.	Cost of installation.
Grain tanks.	From center space.	Sluicing.	24¢	Will handle finer and more colloidal material than above.	Great capacity, low cost.	Cost of installation.
Transfer filter from slime to wash, to discharge compartment, successively, by crane; discharge by air or steam back-pressure.			5 to 22¢	Sticky clayey slime.	Capacity and low cost.	Poor washing and enrichment of wash solutions.
Pulp from wash solutions charged about filter from storage; discharge in same tank by air or steam back pressure.			5 to 22¢	Sticky clayey slime.	Capacity and low cost.	Poor washing and enrichment of wash solutions.
By transfer, much as in preceding, but units small throughout.			?	Sticky clayey slime.	Best washing.	Small units, complicated arrangement
Sand and slime fed on from revolving spreader, wash and removal by similar revolving attachment.			N. D.	Sand and slime together.	Undetermined.	
Filter leaves in tank half full of pulp.	Spray on cake after emerging from slime.	Back pressure of air aided by scraper.	3 to 11¢	Slime of all descriptions.	Simplicity and low cost.	Washing limited.
Similar in all respects to the Butters			?	Sticky clayey slime with low pressures; sandy slime with high pressures.	Higher pressures available for high altitudes.	Inconvenience and expense.
Pulp charged under pressure from centrifugal pump or montejus.	Wash water under pressure.	Basket slid out of cylinder and steam or water discharge used.	20 to 40¢	As above.	As above.	As above.
Pulp charged under pump pressure or by gravity.	Wash water under pressure.	Back pressure of air, lower head opened and filtered slime slides out.	4¢ (gravity).	As above.	As above with apparently very low cost.	?
Pulp charged by gravity, filtration caused by compressed air.	Wash water introduced and forced through cake by air.	End gate opened and cylinder revolved cake loosens slides out.	8¢ (pump)	As above, at El Oro 80% pulp −200 mesh.	As above.	?
Pulp charged to chamber of montejus.	As pulp.	Sluicing.	?	As in other pressure filters.	Higher pressures available.	Probaby cost.

* Net cost of the actual filtering operation is probably less than 3¢. (Between pages 198–199)

When the operation is completed, the cylinder head is removed, a small amount of water admitted and the cylinder revolved; the cake then rolls up like a leaf and is discharged. The standard-size machine as used at El Oro is 42 in. in diameter by 40 ft. long and makes 15 rev. per min. The capacity of this unit is given as 13 charges per 24 hr., each charge being $5\frac{1}{4}$ tons dry weight; the average thickness of the cake being $4\frac{1}{4}$ in., the pressure 38 lb. for forming cake, and 48 lb. for washing, with 66 per cent of the pulp finer than 200 mesh.

*The Sweetland filter-press,** so-called, is in reality a pressure-filter, each leaf being suspended in a frame of the filter-press pattern, these frames being clamped together as usual. Discharge is provided for by means of sluicing-nozzles which wash the spent cakes out of the chamber, as they drop from the leaves when the pressure is released.

SUMMARY AND COMPARISON OF FILTERS

The outstanding features of each of the types previously discussed are collected for convenience in Table XIV.

The cycles of operation of the various types are shown in Fig. 97, part of which is a reproduction of the scheme used by Prof. Young† in his admirable paper. The balance is an extension of the same idea to the filters not discussed by him.

Of the most prominent filtering devices in present use, *viz.:* the Moore-Butters, the Merrill, the Oliver, and the Burt (to which should be added the Ridgeway, though its use on the western Continent is rare) each has distinct merits of its own. Probably they will all survive, and each will eventually find its own field as indeed they are, in some measure doing, at the present time.

The Moore-Butters and the Burt systems seem equally well adapted to the treatment of slimy pulp. Both are open to the objection of discontinuous operation.

The use of the Merrill system, ingenious and clever as it is, seems to be limited to a fairly granular material. It also is of the intermittent type.

The Oliver and the Ridgeway have the great advantage of thin cake and continuous operation.

* Pac. Min., Vol. 16, page 86.

† Slime Filtration, Geo. J. Young, Trans. A.I.M.E., Vol. 42, pages 752–784.

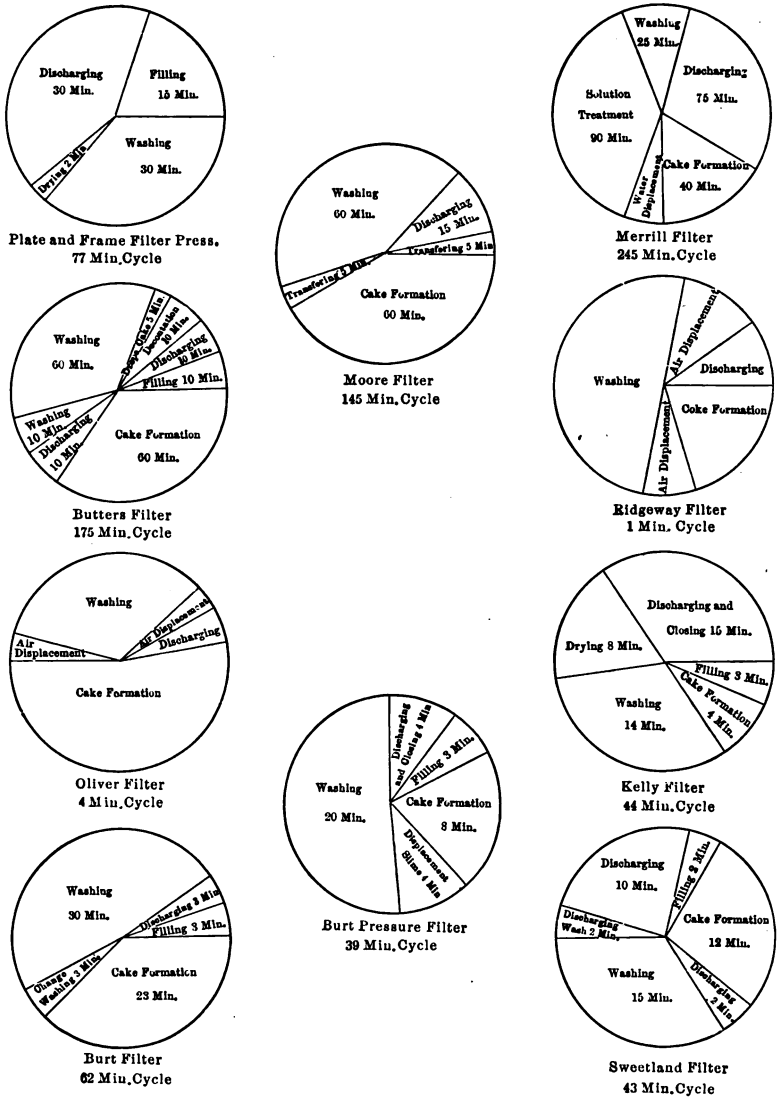


FIG. 97. Cycle of Operation of Various Filters.

The next step of an important character in the development of filter-apparatus for cyanide work will probably be the introduction of a centrifugal filtering device.

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CHAPTER XII

PRECIPITATION OF GOLD AND SILVER

Three methods for the removal of gold and of silver from cyanide solution are in common use:

1. *Precipitation with zinc-shaving.*
2. *Precipitation with zinc-dust.*
3. *Precipitation with an electric current.*

(1) **Precipitation with zinc-shaving** was first successfully used by MacArthur and Forrest. Zinc-plate had been tried without success, and it was not until these inventors conceived the idea of cutting the zinc into fine ribbon-like shavings, that any thoroughly successful precipitation method had been put forward. The object to be attained is, of course, to bring the pregnant solution in contact with the zinc surface, allowing contact to continue for the

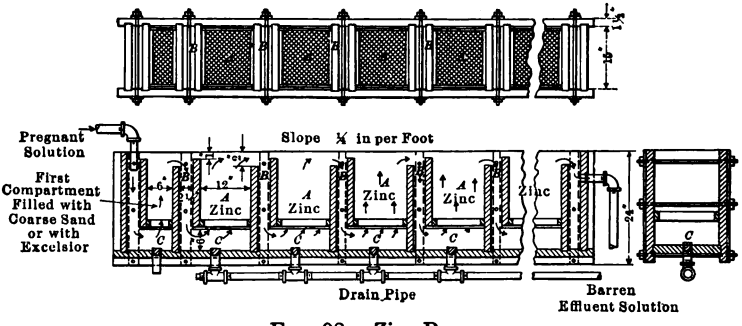

FIG. 98. Zinc Box.

minimum time consistent with commercially complete precipitation. This is usually accomplished by placing the zinc-shaving in compartments in a rectangular trough as shown at *A*, Fig. 98. The solution slowly rises through the wide compartments and overflows at the lower end of each wide compartment into the narrow compartment *B*, through which it flows downward to ascend in the next wide compartment, and so on to the end of the trough, which is set on a slight slope toward the discharge end.

In each zinc-compartment there is a removable tray or false bottom to support the shaving, to assist in maintaining a uniform flow of solution, and to afford opportunity for settlement of precipitate, which may be withdrawn through the drain pipe along the bottom of the box by pulling out the plugs *CC*.

The dimensions and number of the zinc-boxes are, of course, dependent upon the amount and character of the solution to be handled, varying from 12 to 24 ft. in length, and from 18 in. to 3 ft. square in cross-section. They are usually of wood, although iron boxes are not uncommon. Separate cylindrical or rectangular compartments are occasionally used (see Fig. 99); each of these has a projecting lip from which the solution discharges into the descending compartment of the next box in the series. The zinc

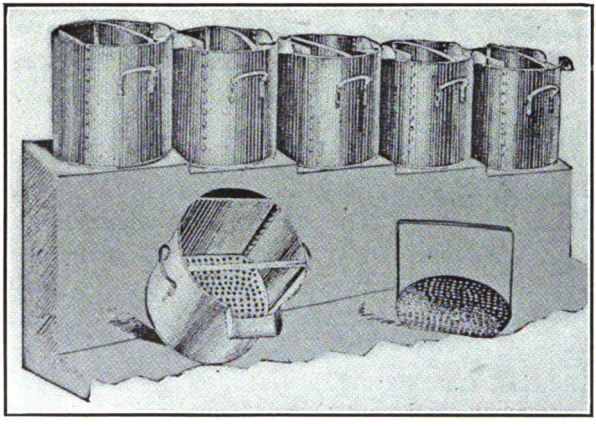

FIG. 99. Separate Zinc Boxes.

used may be prepared at the plant by winding sheet-zinc on a mandrel mounted in a small but substantial lathe, and cutting spirals from this with a side-cutting tool; or if preferred, zinc-shaving may be purchased ready cut. With a large plant the first method is preferable. The thinner the shaving, the greater is the area of zinc surface exposed; at the same time extremely thin shavings break up more readily and foul the precipitated gold and silver. From $\frac{1}{32}$ to $\frac{1}{8}$ in. wide and from $\frac{1}{400}$ to $\frac{1}{1200}$ in. thick, are the common dimensions of cross-section. Uniformity of thickness is an important aid in minimizing the amount of 'short-zinc.' The shaving is loosely but evenly packed into each zinc-compartment, leaving neither open channels nor tightly packed portions

to accelerate or retard solution-flow through various portions of the mass. In boxes containing six or eight compartments it is common to omit the zinc from the compartments at each end. The first box is left empty, or filled with excelsior or some such material, to catch any slime or sediment from the leaching-tanks or filters; and the last box serves as a trap, to prevent detached precipitate from being carried out with the solution. The capacity of any zinc-box, in tons of cyanide solution per day, is a somewhat indeterminate factor, depending upon the strength of the solution, its value, and the amount of foreign metals present. From 1 to 2 tons of solution for each cubic foot of actual zinc-shaving would usually be a conservative figure. The efficiency, and frequently the capacity for precipitation, may be materially increased by dipping the shaving, before use, into a solution of lead acetate or other soluble lead salt. The lead thus precipitated forms with the zinc a galvanic couple that is often an extremely effective precipitant. Much the same purpose may be served by a mixture of zinc and lead shavings or by the use of zinc containing a small amount of lead. The precipitated gold and silver should be loosely adherent to the zinc and of a dark-brownish color.

The development of 'channels' in the zinc-boxes should be watched for and prevented, as should also the polarization of the zinc by hydrogen bubbles. These bubbles may be removed by shaking the zinc. The accumulation of 'white-precipitate' on the shaving is a common occurrence; this may frequently be remedied by temporarily strengthening the solution by means of a strong-solution 'drip,' or by placing lumps of cyanide in the head boxes. Occasionally, white precipitates which do not yield to this treatment are encountered; in such cases the only thing to do is to investigate the substance and find a solvent for it which can be added without detriment to the cyanide solution, or else to let the material accumulate and clean it out periodically.

Precipitation with zinc-shaving has the advantage of being simple in operation, and the apparatus is simple to erect. It is in consequence almost invariably used in small plants in preference to any other method.

(2) **Precipitation with zinc-dust**, although proposed quite early in the development of the cyanide process, and used in several small plants, notably at Deloro, Canada, and at Marysville, Montana, did not come into general use until a few years ago. At the present

time it bids fair to replace the zinc-shaving method in all large plants.

It is evident that the more finely divided any substance, the greater the surface it will expose and the more rapid will be any chemical reaction dependent upon it. The manifest advantages of zinc-dust in this regard were early recognized, but the difficulty of feeding the zinc, and of separating precipitate and zinc from the solution, prevented its general application. Credit for the satisfactory solution of both these problems must be given to C. W. Merrill, although other zinc-dust precipitation plants, evading his patents, are in successful use. The zinc used is a distillation product. According to W. J. Sharwood * it should conform to the following specifications:

"It should be dry and fine; nearly all, say 95 per cent, should pass a sieve of 200 meshes to the linear inch (aperture 0.003) while very little, say not over 1 or 2 per cent, should remain on 100-mesh. . . . It should not show any signs of caking, or contain lumps which do not break up at once when shaken on a sieve. . . . The presence of a little lead, say 2 or 3 per cent, is an advantage. . . . The presence of a little zinc oxide is apparently without detrimental effect, except as diminishing the percentage of active metal. . . . A number of samples which have proved satisfactory in actual use have shown from 85 to over 95 per cent metallic zinc (*i.e.*, zinc in the metallic state), most of them over 90 per cent."

A recent proposal is that zinc (and lead in suitable proportion) shall be atomized at the mill, thereby lowering transportation cost and ensuring a fresh unoxidized precipitant, the composition of which is at the control of the metallurgist.

In the Merrill process (see Fig. 100) the zinc dust is spread upon the belt of an automatic feeder, which is arranged by means of a float and counterweight to travel at a speed proportionate to the volume of solution treated. The dust drops into a small agitating or mixing cone where it is emulsified with air and cyanide solution, and overflows into the suction of a triplex pump, which draws its supply from the gold-tanks † of the cyanide plant.

The mixed stream of pregnant solution and zinc emulsion passes up the discharge column and into the filter-press, where the precious metals and any excess of zinc remain. The clear barren

* J. C. & M. Soc. S. A., Feb., 1912; abs. in Min. & Sci. Press, Vol. 104, page 659; Eng. & Min. Jour., Vol. 93, page 943.

† The misnomer 'gold-tank' is applied to the reservoir for the storage of the pregnant or gold-bearing solution before it goes to the precipitation plant.

solution passes through the pores of the filter-cloth and into the storage-tank. In this process fresh zinc is constantly being introduced, the cloths in the filter-press are coated with a layer of fine zinc, and no particle of solution can pass away without coming into precipitating-contact. The press used in the Merrill system is triangular in section but otherwise is substantially the same as

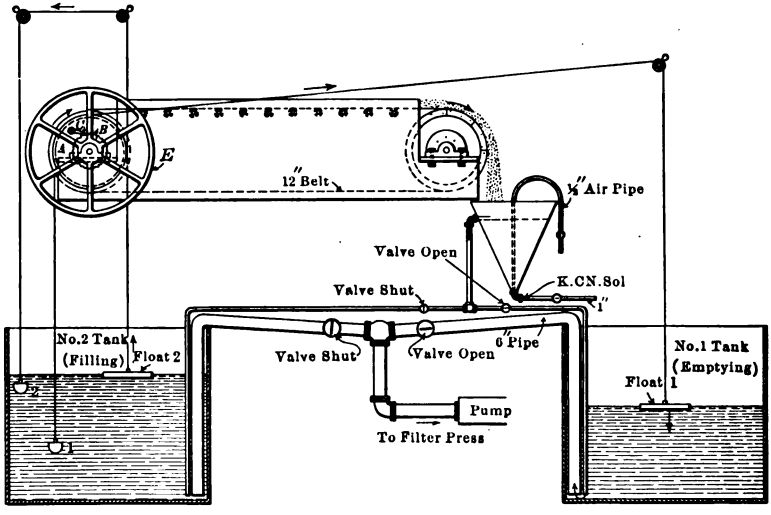

FIG. 100. Merrill Zinc-dust Feeder.

the other standard filter-presses. Interesting variations have been developed both in precipitation and in recovery of the precipitate from this method.

A tiny tube-mill charged with zinc (in one plant aluminium) granules, through which the pregnant solution flows, has the advantage of continually exposing new surfaces of zinc to the solution, at the same time abrading the gold-silver precipitate as soon as it is formed. The precipitate is then carried along by the effluent solution to be deposited in the filter.

At Lluvia de Oro * a simple scheme of double canvas bags attached to a solution header is used to supplement the usual recovery of precipitate by filter-pressing. The bags, when full, are removed from the circuit and others are substituted. Burning of the bag in the precipitate-melting furnace avoids all possibility of loss.

The main advantages of zinc-dust precipitation are the com-

* H. R. Conklin, Eng. & Min. Jour., Vol. 95, page 1001.

pactness of the plant; the low cost for labor; and the absolute security of the precipitate, it remaining locked in the press which is opened only on clean-up days. The recovery of gold and silver is, if anything, more nearly complete than by the use of zinc-shaving, and there is less trouble from the precipitation of base metals interfering with gold recovery, since fresh zinc surfaces are constantly exposed to the incoming solution. The base metals are also more completely removed, thereby improving the tone of the solution. Exact comparison of cost between the zinc-dust and the zinc-shaving methods of precipitation is not possible. Dust can be purchased more cheaply than shaving, but this is offset by the greater operating cost in the zinc-dust precipitation plants. Figures available show a variation from 1.13 to 16 cents per ton of ore for precipitation with dust, and from 1.7 to 20 cents for precipitation with shaving, the conditions in each case not being dissimiliar.

(3) **Precipitation with an Electric Current. Electrolytic precipitation** — using the term to describe deposition by the aid of an electric current from an external source — is an older process than either of the methods that have been discussed. It is, of course, in principle merely the extension of the electroplating process so long in use; nevertheless, it has, in the United States at least, never been popular, and there is only one American plant of consequence using the method at the present time.

Contemporaneous with the introduction of MacArthur-Forrest's zinc-shaving method into South Africa, Siemens and Halske, two German chemists, brought out a process, which, with various modifications, is the most successful electrolytic precipitation method to-day.

In the original Siemens-Halske method, iron anodes and lead cathodes were employed. The iron anode-plates were enclosed in canvas to prevent short circuiting and to collect the iron oxide formed during the electrolysis; the cathodes, in the form of thin sheet lead, were placed between the anodes, and at a distance of about 3 in. from them. The area of the anode and cathode sheets is merely a matter of convenience, from 4 to 6 sq. ft. being common. The iron anodes are from $\frac{1}{16}$ to $\frac{1}{8}$ in. thick and the lead cathodes from 0.0032 to 0.01 in.; from 0.04 to 0.06 amp. per sq. ft. of anode are used, at a pressure of 5 to 10 volts. The immense area of lead foil required soon led to cutting the

foil into narrow strips, a few inches wide, and corrugating them in the direction of their width. Under favorable circumstances it was found that from 15 to 20 cu. ft. of tank volume were necessary for each ton of solution handled per 24 hours. By increasing the density of the current, and by using lead peroxide anodes in place of iron, Charles Butters has been able to reduce materially the volume of tankage required.

The gold in every case, of course, is deposited as an adherent film on the lead-foil cathode.

More recently Butters, at his Virginia City plant, and at Minas Prietas, in Mexico, has substituted sheets of commercial tin-plate for the lead cathodes. The gold and silver do not adhere to the tin as they do to the lead, and may be rubbed off from time to time to accumulate, as a sludge or slime, in the bottom of the tank. In spite, however, of many ingenious improvements in electrical precipitation, it appears to be necessary, in almost every case, to send the solution to zinc-boxes for removal of the last extractable portions of gold and silver. The process can, therefore, in no case be regarded as a final one, and in view of the simplicity of zinc-boxes for small plants, and the convenience and satisfactory work of zinc-dust in the larger plants, it is hard to see how electrolytic precipitation in its present form is likely to become at all general in its application.

In the treatment of solutions from certain complex ores, however, electrolytic precipitation will probably continue to hold its own, because of its ability to remove objectionable metals from the solution, and thereby regenerate and make available the cyanide locked up and otherwise useless.

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CHAPTER XIII

RECOVERY AND TREATMENT OF PRECIPITATE

The method of handling the precipitated gold and silver will, of course, be largely dependent upon the method of precipitation employed.

With zinc-box precipitate, in order that the deposit adhering to the zinc may be thoroughly loosened, it is customary to strengthen the cyanide solution materially for a few hours before a clean-up is made. This done, the cyanide is displaced by clear water, to which alum may be added to facilitate settlement. Then beginning at the head box, the zinc is lifted out by hand and carefully sprayed to remove adhering precipitate; this may be done over the zinc-box or over a special clean-up tank. The cleaned zinc is then laid aside in water, and the next compartment is treated in the same way till all coarse zinc has been removed. There will then remain below the trays in the boxes the residue to be treated, consisting of the so-called 'short-zinc' (pieces large enough to be removed by screening), various compounds of zinc, gold (or silver), and cyanide, and the bulk of the precipitated gold, together with any other metals or substances which might have been precipitated or deposited on the zinc. This material, together with the water remaining in the boxes after siphoning off the clear liquid, is either drawn out through spigots in the bottom of the boxes or bailed out with wooden or enameled buckets, and is placed in a clean-up tank, where the precipitate is given sufficient time to settle thoroughly; the supernatant liquid is then carefully siphoned away. In some works the muddy precipitate is run through a 60-mesh screen by which means much short-zinc is at once separated. In other cases the material from the boxes goes direct to a small filter-press. In any event the precipitate, either before or after drying, should be freed as much as possible from short-zinc. When this has been done we have a finely-divided product of uncertain composition, but carrying a considerable proportion of zinc and other foreign substances, which must be removed before a satisfactory bullion can be made.

Where zinc-dust is the precipitant, a finely-divided slime is recovered in the presses, which corresponds to the screened product from the boxes, except that it is likely to contain more zinc. The electrolytic precipitate is usually cleaner than that from either of the other processes.

In many plants the finely-divided precipitate, whatever its source, is carefully sampled, sacked, and shipped by express to the smelter, the cost of refining at the plant being greater than express plus treatment at the smelter.

In cases, however, where treatment of the precipitate at the mill is desirable, several methods are available. Of these the ones in most common use will be described:

I. *Acid treatment for removal of zinc, subsequent drying and melting the residue with suitable fluxes.*

II. *Roasting, with or without niter, to remove zinc; with treatment of residue similar to the preceding.*

III. *Direct smelting with an oxidizing charge to flux the zinc and other impurities, the resulting bullion (usually lead) being refined for gold and silver.*

I. The Acid Treatment is the one most commonly used in the United States. The precipitate, after being transferred to a lead-lined tank, is treated with sufficient 10 per cent sulphuric acid to dissolve all the zinc, arrangements being made for heating and stirring the solution if necessary. This operation should be carried out under a hood on account of the fumes evolved. Hydrochloric acid is sometimes used in place of sulphuric. When all the zinc has been dissolved, the tank may be filled with water, the precipitate allowed to settle, and the zinc sulphate solution decanted by suitable means, although, as before, much more satisfactory results will be attained by the use of a small filter-press. In any event the residue from this treatment will be practically zinc-free and ready for direct fusion. In some plants, however, the precipitate is roasted before melting, but this would seem scarcely necessary in most cases.

Before melting, the required fluxes should be thoroughly mixed with the precipitate. Many of the larger plants briquette the flux-precipitate mixture to prevent dusting during the early stages of fusion. The fluxes to be used vary considerably with the choice of the operator and the nature of the precipitate. From 30 to 50 parts of borax and 9 to 30 parts of bicarbonate of soda, to each 100

parts by weight of precipitate, is a common charge. The melting is usually done in graphite crucibles, in coal, coke, wood, or charcoal furnaces. Recently the use of gasoline or crude-oil as a fuel has become more common and there are on the market several melting furnaces fired by this means, and so arranged to tilt on trunnions (see Fig. 101), that it is unnecessary to remove the crucible from the furnace in pouring. This arrangement is so convenient that its use is bound to become quite general.

II. The Roasting Method, while never very popular in the United States, has been used quite extensively elsewhere. Niter, either in the form of powder, or as a saturated solution, is fre-

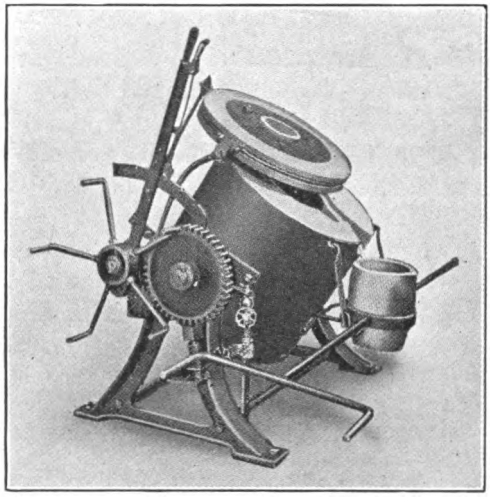

FIG. 101. Tilting-type Melting-Furnace.

quently added to assist in oxidizing the zinc, before drying the precipitate. The roasting is usually done in some form of muffle furnace, the precipitate being placed in roasting pans before it is put in the muffle. The whole operation must be conducted with extreme care to avoid loss by dusting, as well as to minimize the mechanical and chemical losses connected with the volatilization of the zinc.

III. The Process of Direct Smelting is coming into favor in many of the larger mills. Where zinc-shaving is used the short zinc is usually removed by running the zinc-box sludge through a 20- or 40-mesh screen. When this practice is followed, the method

used at the Dos Estrellas mill, as described by Walter Neal,* is simple and efficient. The precipitate, after screening out the 'zinc-shorts,' is run to a filter-press, dried with compressed air, and discharged into a steam-dryer car. The material is handled as little as possible so as not to break the cakes formed in the press. Fluxes are shoveled on top of the precipitate, the dryer car run to the furnace and precipitate and fluxes shoveled into crucibles ready for melting. The charge is made up of:

Precipitate	100 parts.
Borax	15 parts.
Soda bicarbonate	8 parts.
Sand	4 parts.

Scrap iron in excess.

The so-called Tavener process for treating precipitate, resembles the ordinary fire-assay for gold and silver. The precipitate is mixed with litharge, borax (and other fluxes if necessary) and some reducing agent to ensure a lead button and the whole is fused. The lead button is cupelled, and the gold and silver are recovered as a clean bullion. This method is especially popular in South Africa, and a number of the larger mills in the United States are reported as finding it eminently satisfactory.

Another modification of the direct smelting process is in use in some of the larger Nevada mills.† The precipitate is briquetted with fluxes in the following proportions: —

	Parts
Precipitate (dry weight)	100
Litharge	100 to 125
Concentrate (S, 35 per cent, SiO_2, 30 per cent)	60 to 70

Moisture about 9 per cent.

These briquettes are charged into a small blast-furnace with some old slag, cupel bottoms, etc., and smelted for lead bullion, which is later cupeled as in the Tavener process. The results are excellent and the cost is extremely low.

Treatment of Solution after Precipitation. It is well to remember that the precipitation plant yields two products — precipitate and effluent solution. Upon the proper disposal and treatment of the latter depends, to no small extent, the successful and economical treatment of the ore.

* Min. & Sci. Press, Vol. 98, page 327.

† J. W. Hutchinson, Min. & Sci. Press, Vol. 102, page 717.

In standard practice the effluent solutions are run to sump tanks from which they are withdrawn as needed for re-use. These sump solutions are naturally weaker in cyanogen than they were before use, and this loss of cyanide is a serious item of cost; in some plants, indeed, it is the largest single cost-factor. If, for example, the solution, as applied to the ore, shows by titration with silver nitrate a strength equivalent to 0.52 per cent KCy and the same solution when it reaches the sump titrates 0.27 per cent KCy, then we have an apparent consumption of 0.25 per cent KCy, or 5 pounds per ton of solution which at 12 to 18 cents per pound means serious expense from this source alone. It does not necessarily follow, however, that this amount of cyanide has been irrevocably lost, the greater part of the consumption being due, not to destruction of cyanide, but to the formation of cyanides of zinc in the extractor-boxes, of cyanides of iron from the tanks and pipes, and from particles of iron abraded from crushing machines, and cyanides of various other base metals according to the character of the ore undergoing treatment. Ferrocyanides, sulphocyanides, cyanates, and numerous other compounds are also likely to be formed under certain conditions.

If the ore is a comparatively simple one, continued re-use of the solution for long periods may cause no appreciable effect either upon solution or precipitation. With the more complex ores, however, especially those containing base metals in soluble form, the accumulation of metallic cyanides has an effect both on solution and on precipitation, although usually this effect is more marked upon the latter than upon the former. In certain cases, however, as cited by Julian and Smart,* solutions which have been used have no longer the same dissolving power although raised to the original cyanide strength. These difficulties may in general be solved in one of the following ways:

(1) Through keeping the solution slightly alkaline by the addition of lime to the ore during crushing, or by addition of 'milk of lime' or caustic soda to the cyanide solution itself. This, indeed, is necessary in all successful cyanide work. The proper amount of protective alkali (*i.e.*, alkali over and above that of the cyanide itself) is a matter of determination for each particular ore. An interesting case is cited by L. McN. B. Bullock,† in which, by an

* Cyaniding Gold and Silver Ores, 1st Ed., pages 71–72.

† Min. & Sci. Press, Vol. 94, page 719.

increase of alkalinity from 0.04 to 0.14 per cent (caustic soda), the cyanide consumption was cut from 4.5 to 1.5 pounds per ton of ore.

(2) Through complete aeration of the solutions, as practiced by some of the silver cyanide mills in which the sumps solutions are kept constantly circulating through cones by the aid of centrifugal pumps. In this way various reducing agents collected from the ore, and which would affect the solvent action of the solution, are rendered innocuous.

(3) Through removal of the base metal cyanides by acid precipitation.*

$$K_2Cu_2Cy_4 + H_2SO_4 = K_2SO_4 + Cu_2Cy_2 + 2\,HCy.$$

If the supernatant liquid be decanted from the Cu_2Cy_2 precipitate, and run into a tank containing an excess of lime water we shall have:

$$K_2SO_4 + 2\,HCy + Ca(OH)_2 = CaSO_4 + 2\,KCy + 2\,H_2O.$$

In this manner half of the cyanogen locked up may be recovered. It is possible also to recover a large portion of the cyanogen from the cupric cyanide precipitate.†

Various methods, such as the addition of ferric salts for the precipitation of ferrocyanides as prussian blue, or of alkaline sulphides to decompose cyanides of various metals, have been proposed and used to some extent. Electrolysis has recently been given a thorough trial, as a means of cyanide regeneration ‡ and has been reported ineffective.

In general the first two methods mentioned must be relied upon for everyday work. Acid regeneration may be advisable, but should only be used in extreme cases.

* S. B. Christy, Trans. A.I.M.E., Vol. 26, page 747.

† R. P. Wheelock, Min. & Sci. Press, Vol. 99, page 814.

‡ J. E. Clennell, Eng. & Min. Jour., Vol. 91, page 1064.

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PART III

THE TREATMENT OF GOLD AND OF SILVER ORES

This portion of the text is intended to correlate the various processes which have so far been described, and to point out the relations and functions of the various parts of the modern gold- and silver-milling plants.

CHAPTER XIV

TREATMENT OF GOLD ORES

It will be at once accepted as axiomatic, that in order for the particles of gold existing in an ore to be removed from it, they must be exposed to the action of the particular process designed for their recovery.

This may be accomplished in one of three ways: (1) By removing from the gold particles all of the enclosing envelope of rock; (2) by removing enough of the envelope, to enable the reagent to come in sufficient contact with the gold, either to hold the gold and its matrix, or to remove it therefrom by solution or otherwise; (3) by rendering the ore so porous, that to all intents and purposes the gold particle is as completely liberated as though the surrounding gangue material had been entirely removed.

The first two results may be brought about as the result of crushing, the completeness of liberation being a function of the size of the gold particles and of the character and fineness of crushing.

The third condition — in which, owing to the porosity of the matrix, complete liberation is not essential — is brought about usually as the result of natural or artificial oxidation of other minerals (usually sulphides) existing as a part of the gangue. Examples are seen in the oxidized, porous, and cellular out-crops of sulphide ore-bodies. Much the same result may be brought about by roasting sulphide ores under suitable conditions.

All this presumes that the gold exists in the native state, and free from adhering films, and that it may hence be released from its matrix, much as the kernel of a nut may be set free by cracking the shell.

Should the gold not exist in the native state but in combination with some other element, then it is still true that the gold compound must be similarly liberated from its envelope before any mechanical or chemical extraction can take place. This brings us to the point of recognizing crushing as the *sine qua non* of any metallurgical process which is not planned, as is smelting for instance, to decompose the rock-mass as a whole.

Having the particles liberated and exposed, there then remains to be decided the method of removing the gold or gold mineral from the rest of the ore.

If the gold is coarse, free, in the sense of being native, and free from adhering films of oxide or other substance, then it is evident that it should be recovered as far as possible by amalgamation. In fact, we might go farther than this and say, that all gold which can be recovered by ordinary methods of amalgamation, should be so saved, and that only that portion of the precious metal not effectively and readily recoverable by this means should be subjected to cyaniding. The reasons for this are obvious enough. Coarse gold particles dissolve but slowly in cyanide-solution, thus increasing materially the time required for treatment; secondly, complete precipitation and complete recovery of solution are alike impossible, so that in no case will all of the gold particle find its way into the cyanide clean-up.

With regard to concentration the case is different. Concentration is not a final process. The product recovered by this means must be subjected to still further treatment for the recovery of any gold it may contain, and unless by concentration we can remove substances, which if left in the ore would be detrimental to cyaniding or, unless the gold recovered in this way cannot be recovered by direct cyaniding, concentration should be omitted.

In the treatment of gold ores, then, we have the following possibilities:

(1) *Amalgamation.*
(2) *Amalgamation and concentration.*
(3) *Amalgamation and cyanidation.*
(4) *Amalgamation, concentration, and cyanidation.*
(5) *Cyanidation.*

(1) **Amalgamation**, as a sole means of recovery, is rarely, if ever, sufficient. In a few cases, with exceptional ores in which the gold is clean and coarse, and the conditions are such that the ore is completely oxidized, it may be found that practically all of the gold is recoverable by this means, but such conditions are decidedly rare, and at best can last only until the level of oxidation is bottomed. Such ores are the ones referred to as 'free-milling.'

(2) **Amalgamation and Concentration.** Prior to the advent of the cyanide process, this was the combination almost universally employed for such ores as we are discussing. Good recoveries

were made, and immense tonnages treated by this means, and the process has by no means become obsolete. Stamps in this field have been particularly successful in the combined function of crushers and amalgamators, the plate tailing in such cases being sent to vanners or, in the more recent plants, after classification, to Wilfley-type tables and to vanners. The concentrate is treated on the ground by cyanidation or chlorination, or shipped to smelting works.

(3) **Amalgamation and Cyanidation** is proving itself the most rational mode of treatment for an immense tonnage of ore at the present day. Where, as previously remarked, the omission of concentration does not cause excessive cyanide consumption, and where the net recovery by cyanide treatment is as high as by concentration and shipment of the concentrate, allowing for the extra cost of the latter method, then, of course, entirely local treatment is to be preferred. It should be noted in this connection, that while gold in the form of bullion is paid for at the full value of $20.67 per ounce, that existing in the concentrate will, when shipped to a smelter, bring only $19 or $19.50 unless the concentrate is particularly high grade, in which case as much as $20 per ounce may be realized.

(4) **Amalgamation, Concentration, and Cyanidation** together must necessarily be employed in the treatment of the more complex ores, and are of course, frequently seen in older plants to which cyaniding has been added as an after-thought. Such a combination means great complexity of plant with high treatment cost, and is avoided wherever it is possible to omit one of the steps. It seems probable that, as time goes on, and we are able to handle the sulphide minerals more successfully in the cyanide plant, concentration will become a less important factor in the treatment of gold-ores than is at present the case. Much the same thing may be said of concentration followed by cyaniding without the use of amalgamation.

(5) **Cyanidation.** Direct and unassisted cyanidation is finding many advocates, especially in South Africa, but this matter has already been discussed quite fully. Undoubtedly, there are certain ores where the percentage of gold recoverable by amalgamation is so small, that even the slight extra expense of passing the pulp over copper-plates and recovering the amalgam from them is a matter of doubtful economy. On the other hand, it is a regrettable fact that less attention is paid to amalgamation in

many cyanide mills than would have been tolerated under the system in vogue before cyaniding reached its present efficiency. In fact, there are cases where there is reason to think that amalgamation is slighted, to give the cyanide plant a chance to make a better showing. This, of course, is the height of folly.

In the preceding discussion no distinction has been drawn, in cyanidation, between sand and slime treatment,* nor have other adjuncts, such as roasting or solution-oxidizers, been mentioned.

It is now proposed to show, in detail, typical plants illustrative of good practice under each of the above conditions. It is not necessarily to be assumed that the process in each case is best suited to the particular ore, nor is it true that the mill arrangements and details of treatment are in every case such as would be approved by a jury of competent metallurgists.

In the main, however, the plants chosen represent standard practice and the reader will find many of the best plants in the world included in the following diagrams.

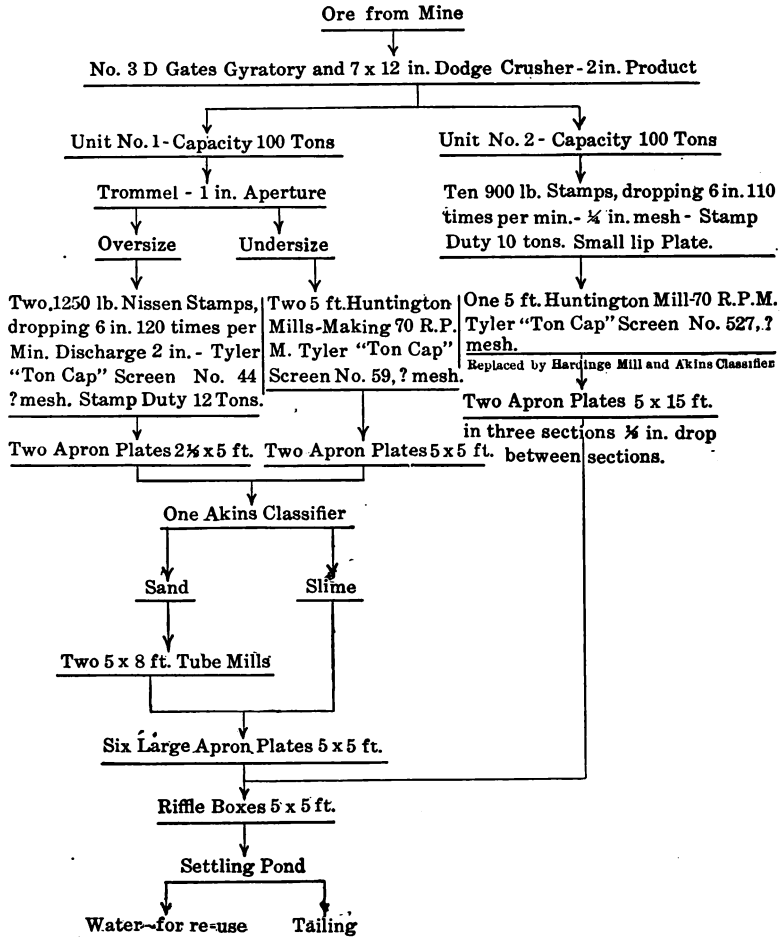
* As used in ordinary mill-parlance, 'sand' is a product which is treated by leaching; 'slime' is a product which is not or cannot be so handled. It would generally be more exact to say, 'leachable product' and 'non-leachable product,' than to use the terms 'sand' and 'slime' in their usual significance.

(1) AMALGAMATION

Examples under this class are rare and I can recall but one plant, that of the Round Mountain Mining Company at Round Mountain, Nevada. The process there used is shown by the following diagram.

Character of Ore: — The gold is associated with iron oxide lying in seams in a rhyolitic rock. Average value $7.00 per ton. The ore would be classified as soft.

Flow Sheet:


Ore from Mine
No. 3 D Gates Gyratory and 7 x 12 in. Dodge Crusher - 2 in. Product
Unit No. 1 - Capacity 100 Tons
Unit No. 2 - Capacity 100 Tons
Trommel - 1 in. Aperture
Oversize
Undersize
Ten 900 lb. Stamps, dropping 6 in. 110 times per min. - ¼ in. mesh - Stamp Duty 10 tons. Small lip Plate.
Two 1250 lb. Nissen Stamps, dropping 6 in. 120 times per Min. Discharge 2 in. - Tyler "Ton Cap" Screen No. 44 ? mesh. Stamp Duty 12 Tons.
Two 5 ft. Huntington Mills-Making 70 R.P. M. Tyler "Ton Cap" Screen No. 59, ? mesh.
One 5 ft. Huntington Mill-70 R.P.M. Tyler "Ton Cap" Screen No. 527, ? mesh.
Replaced by Hardinge Mill and Akins Classifier
Two Apron Plates 5 x 15 ft. in three sections ½ in. drop between sections.
Two Apron Plates 2½ x 5 ft.
Two Apron Plates 5 x 5 ft.
One Akins Classifier
Sand
Slime
Two 5 x 8 ft. Tube Mills
Six Large Apron Plates 5 x 5 ft.
Riffle Boxes 5 x 5 ft.
Settling Pond
Water for re-use
Tailing

Results: — Saving 88 per cent. Cost of milling, including 20¢ for depreciation, is $1.00 per ton.

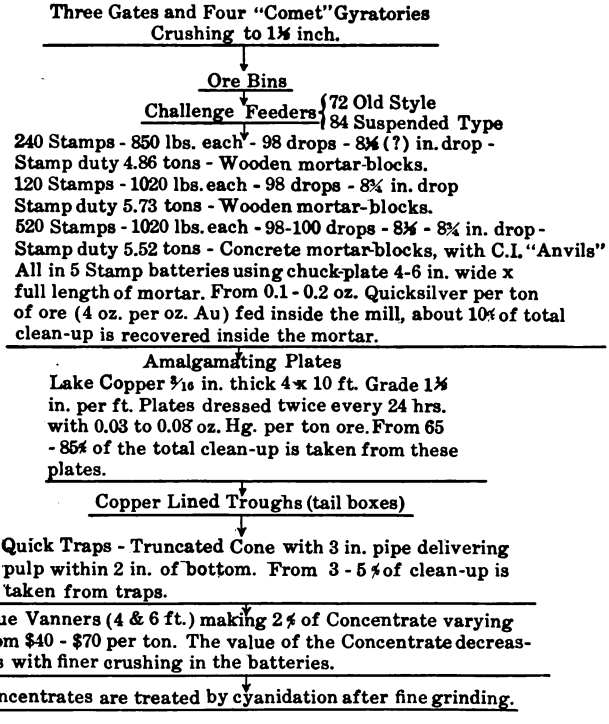
FIG. 102.

(2) AMALGAMATION AND CONCENTRATION

An excellent example of this practice is afforded by the immense mills of the Alaska-Treadwell Gold Mining Company.* The following diagram shows the methods there used.

Character of Ore: — Altered syenite carrying free gold, auriferous pyrite and arsenopyrite. Average value of ore $2.59 per ton (in 1912).

Flow Sheet:


Three Gates and Four "Comet" Gyratories
Crushing to 1½ inch.
Ore Bins
Challenge Feeders
72 Old Style
84 Suspended Type
240 Stamps - 850 lbs. each - 98 drops - 8¾ (?) in. drop -
Stamp duty 4.86 tons - Wooden mortar-blocks.
120 Stamps - 1020 lbs. each - 98 drops - 8¾ in. drop
Stamp duty 5.73 tons - Wooden mortar-blocks.
520 Stamps - 1020 lbs. each - 98-100 drops - 8½ - 8¾ in. drop -
Stamp duty 5.52 tons - Concrete mortar-blocks, with C.I. "Anvils"
All in 5 Stamp batteries using chuck-plate 4-6 in. wide x
full length of mortar. From 0.1 - 0.2 oz. Quicksilver per ton
of ore (4 oz. per oz. Au) fed inside the mill, about 10% of total
clean-up is recovered inside the mortar.
Amalgamating Plates
Lake Copper 5/16 in. thick 4 x 10 ft. Grade 1½
in. per ft. Plates dressed twice every 24 hrs.
with 0.03 to 0.08 oz. Hg. per ton ore. From 65
- 85% of the total clean-up is taken from these
plates.
Copper Lined Troughs (tail boxes)
Quick Traps - Truncated Cone with 3 in. pipe delivering
pulp within 2 in. of bottom. From 3 - 5 % of clean-up is
taken from traps.
Frue Vanners (4 & 6 ft.) making 2 % of Concentrate varying
from $40 - $70 per ton. The value of the Concentrate decreas-
es with finer crushing in the batteries.
Concentrates are treated by cyanidation after fine grinding.

Results: — Average recovery is about 95 per cent. This recovery is equally divided between amalgamation and concentration. The total cost of milling is 21.7¢ per ton distributed as follows: crushing 2.1¢; tramming 1.4¢; stamping 13.8¢; concentrating 5.4¢.

FIG. 103.

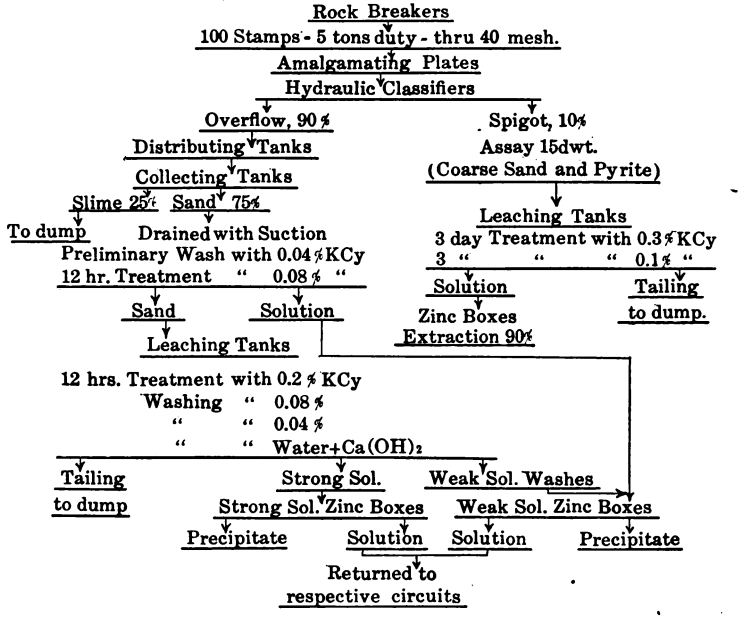
* Robt. A. Kinzie, The Treadwell Mines, Trans. A.I.M.E., Vol. 34, page 365.

(3) AMALGAMATION AND CYANIDATION

I. Sand Treatment. The earlier practice on the South African Rand was essentially of this type.

Character of Ore: — South African "Banket." Gold, free and fine in firmly consolidated conglomerate. Value 42*s.* per ton.

Flow-Sheet:



Results: — Average extraction 78 per cent. Milling and cyaniding cost 9*s.* (1894).

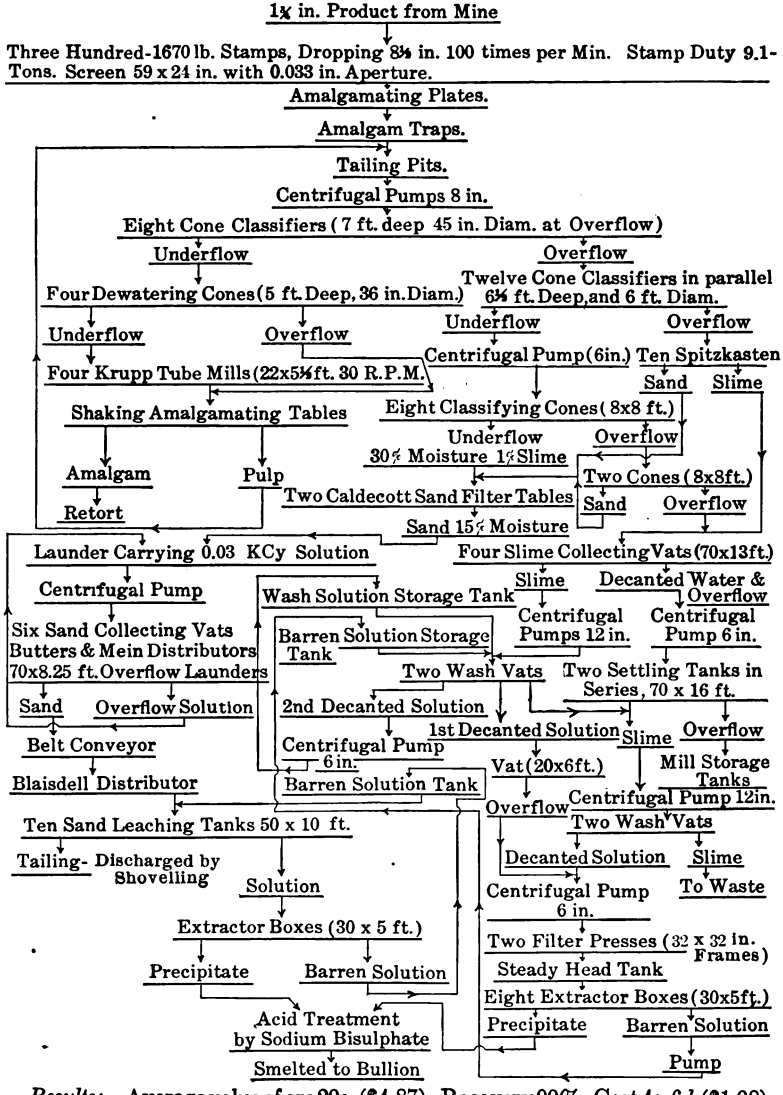
FIG. 104.

(3) AMALGAMATION AND CYANIDATION

II. Treatment of Sand by Leaching and of Slime by Decantation is exemplified by the mill of the Simmer Deep Ltd.* at Johannesburg, S. Africa.

Character of Ore: — Typical South African "Banket," *i.e.*, consolidated conglomerate carrying free gold and pyrite.

Flow Sheet:



Results:—Average value of ore 20s. ($4.87). Recovery 90%. Cost 4s. 6d. ($1.08).

FIG. 105.

The phraseology, it will be noted, is slightly different from that in use in the cyanide plants of the United States.

* Min. Ind., Vol. 18, page 345.

(3) AMALGAMATION AND CYANIDATION

II (*a*). Treatment of Sand by Leaching and of Slime by Filtration. The best example of this type of treatment is probably that afforded by the Pittsburg-Silver Peak Gold Mining Co's. mill at Blair, Nevada.*

Character of Ore: — Quartz and alaskite containing free gold with small amounts of pyrite and galena.

Flow Sheet:

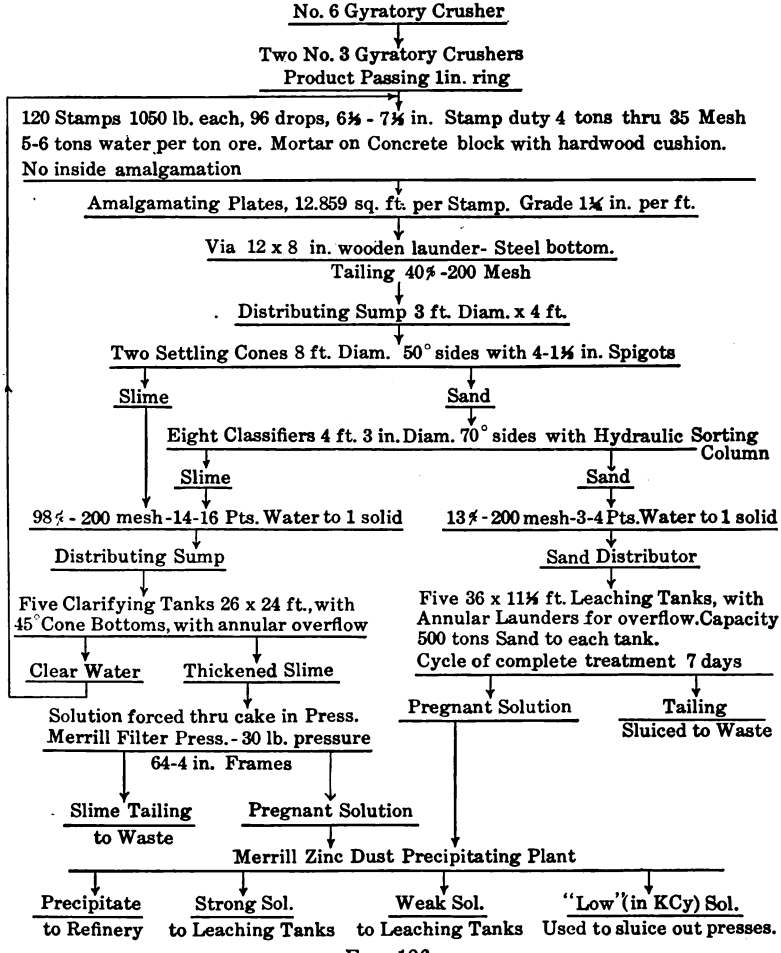

No. 6 Gyratory Crusher
Two No. 3 Gyratory Crushers
Product Passing 1in. ring
120 Stamps 1050 lb. each, 96 drops, 6½ - 7½ in. Stamp duty 4 tons thru 35 Mesh
5-6 tons water per ton ore. Mortar on Concrete block with hardwood cushion.
No inside amalgamation
Amalgamating Plates, 12.859 sq. ft. per Stamp. Grade 1¼ in. per ft.
Via 12 x 8 in. wooden launder- Steel bottom.
Tailing 40% -200 Mesh
Distributing Sump 3 ft. Diam. x 4 ft.
Two Settling Cones 8 ft. Diam. 50° sides with 4-1½ in. Spigots
Slime
Sand
Eight Classifiers 4 ft. 3 in. Diam. 70° sides with Hydraulic Sorting Column
Slime
Sand
98% - 200 mesh - 14-16 Pts. Water to 1 solid
13% - 200 mesh - 3-4 Pts. Water to 1 solid
Distributing Sump
Sand Distributor
Five Clarifying Tanks 26 x 24 ft., with 45° Cone Bottoms, with annular overflow
Five 36 x 11½ ft. Leaching Tanks, with Annular Launders for overflow. Capacity 500 tons Sand to each tank.
Cycle of complete treatment 7 days
Clear Water
Thickened Slime
Pregnant Solution
Tailing
Sluiced to Waste
Solution forced thru cake in Press.
Merrill Filter Press. - 30 lb. pressure
64-4 in. Frames
Slime Tailing
to Waste
Pregnant Solution
Merrill Zinc Dust Precipitating Plant
Precipitate
to Refinery
Strong Sol.
to Leaching Tanks
Weak Sol.
to Leaching Tanks
"Low" (in KCy) Sol.
Used to sluice out presses.

FIG. 106.

* Henry Hanson, Min. & Sci. Press, Vol. 98, page 657; also Min. & Min., Vol. 29, page 569.

Results: — Average value of ore $5.50 per ton. Extraction 92 per cent. Cost $1.04.*

* Distribution of cost:

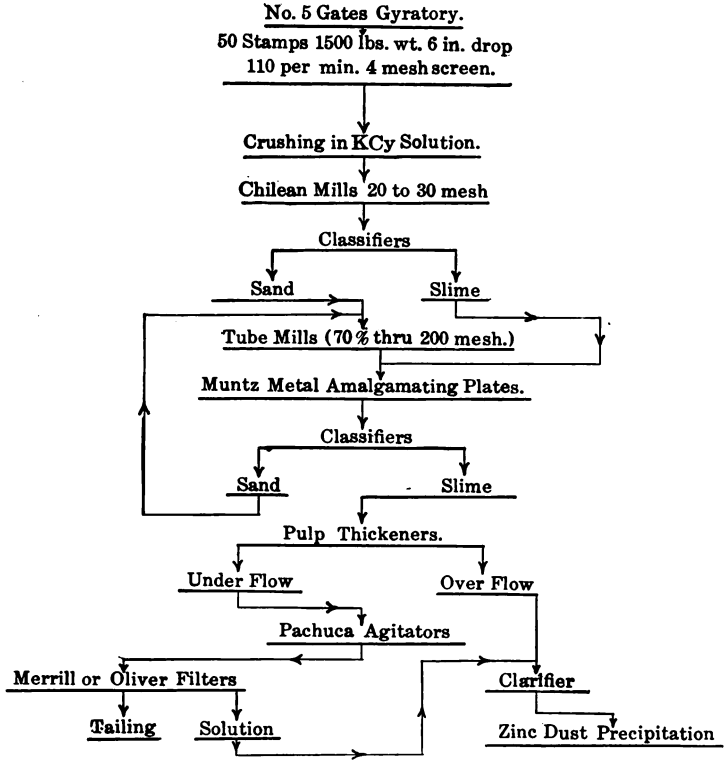
Stamping	$0.298	
Amalgamating	0.047	
Neutralizing and settling	0.074	
Leaching and sluicing	0.145	
Filter-pressing	0.104	
Precipitating	0.036	
Refining	0.048	
Assaying	0.033	
Water service	0.070	
Heating	0.007	
Superintendent and foreman	0.053	
Total direct operating		$0.915
Pro-general	0.079	
Suspense account	0.046	
Total operating		$1.040

(3) AMALGAMATION AND CYANIDATION

III. **'All-slime' Treatment** is exemplified by the flow-sheet below which illustrates a plant designed by the writer for an ore similar to that discussed on page 244.

Character of Ore: — Fine grained cherty quartz with some andesitic country rock — small amount of fine grained complex sulphides.

Flow Sheet:



Results Anticipated: — Average assay of ore 0.10 oz. gold, 10 oz. silver. Extraction 95 per cent gold, 85 per cent silver. Estimated cost with H.P. at $50.00 per annum, $2.00 per ton.

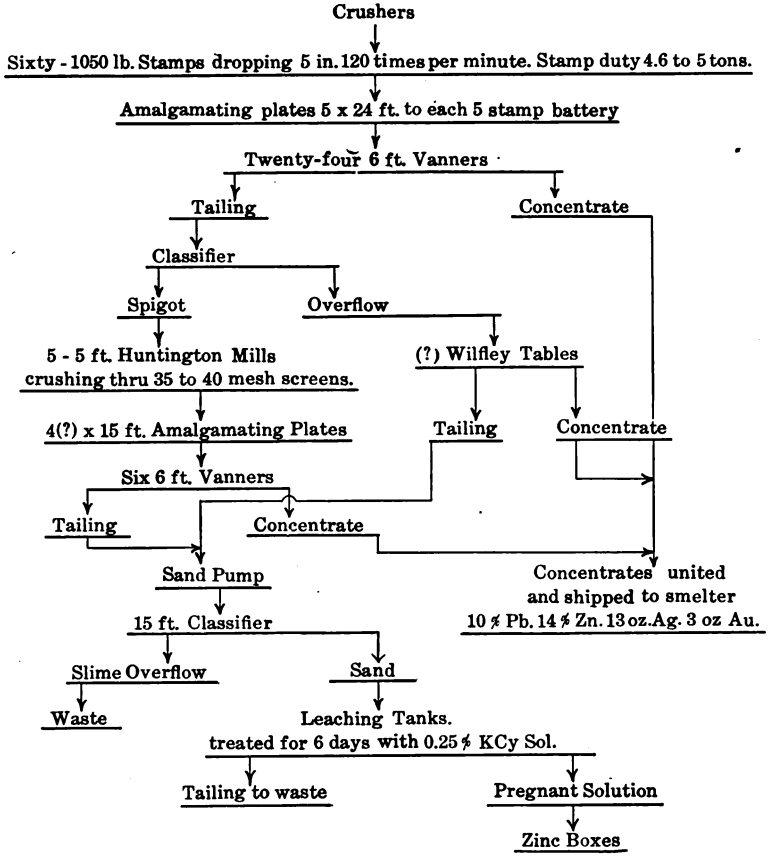
FIG. 107.

(4) AMALGAMATION, CONCENTRATION, AND CYANIDATION

I. Cyanide Treatment of Sands only. This treatment is well exemplified by the mill of the Camp Bird Ltd. near Ouray, Colorado.*

Character of Ore: — The ore is siliceous, carrying free gold, with galena, blende, pyrite, and chalcopyrite. Value $25 per ton, distributed about as follows: gold $23; silver $1; copper and lead $1.

Flow Sheet:



Results: — Extraction 95 per cent of value, distributed as follows: Amalgamation 64 per cent; concentration 29 per cent; cyanidation 7 per cent. Costs, stamping and concentrating $1.12; cyaniding 58¢; shipping and selling $1.77.

FIG. 108.

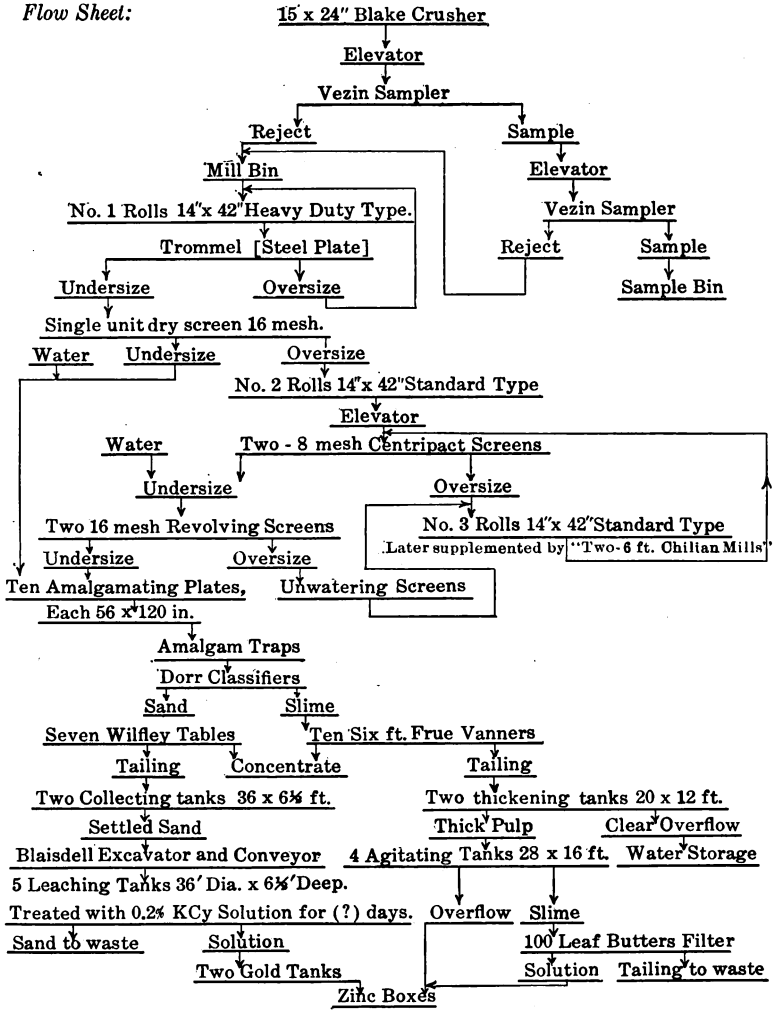
* T. T. Read, Min. & Sci. Press, Vol. 97, page 667; also Min. Ind., Vol. 17, page 439.

(4) AMALGAMATION, CONCENTRATION, AND CYANIDATION

II (*a*). **Cyanide Treatment of both Sand and Slime following Dry Crushing.** An uncommon type; exemplified by the mill of the Bullfrog Water and Reduction Co. (The Montgomery Shoshone mill) at Bullfrog, Nevada.*

Character of Ore: — Shattered and silicified rhyolite, highly altered, containing limonite, calcite and small amount of pyrite.

Flow Sheet:



Results: — Average value of ore $10 to $12 per ton, silver to gold in ratio 12 to 1. Extraction 90 per cent. Cost $2.25 per ton.

FIG. 109.

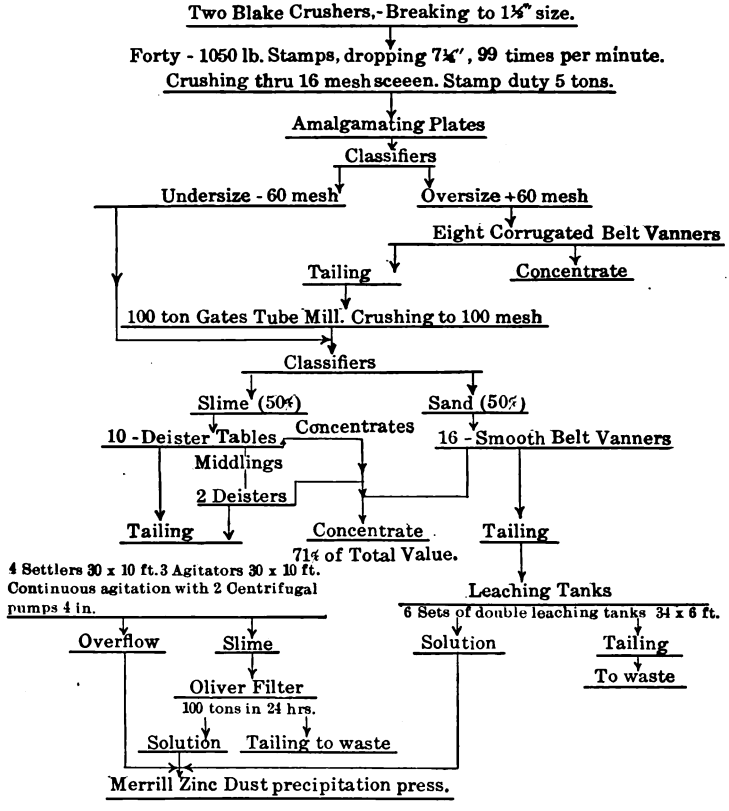
* Enos R. Ayres, Eng. & Min. Jour., Vol. 83, page 376.

(4) AMALGAMATION, CONCENTRATION, AND CYANIDATION

II (*b*). Separate Treatment of Sand and Slime, following Wet Crushing. The mill of the Daly Reduction Co.* at Hedley, B. C., is a good example.

Character of Ore: — Average $12 per ton in value. Arsenopyrite in lime-silicate gangue.

Flow Sheet:



Note. — Amalgamation has been dispensed with, due to changes in character of the ore, but is retained here as an illustration of the type of practice discussed.

Chemicals Used.

9 lbs. lime (mainly for settling).	¾ lb. sulphuric acid.
½ lb. zinc dust.	1½ lb. cyanide.

Results: — Total extraction 93 per cent. Cost of mining and milling $6 per ton, — 5000 tons a month treated.

FIG. 110.

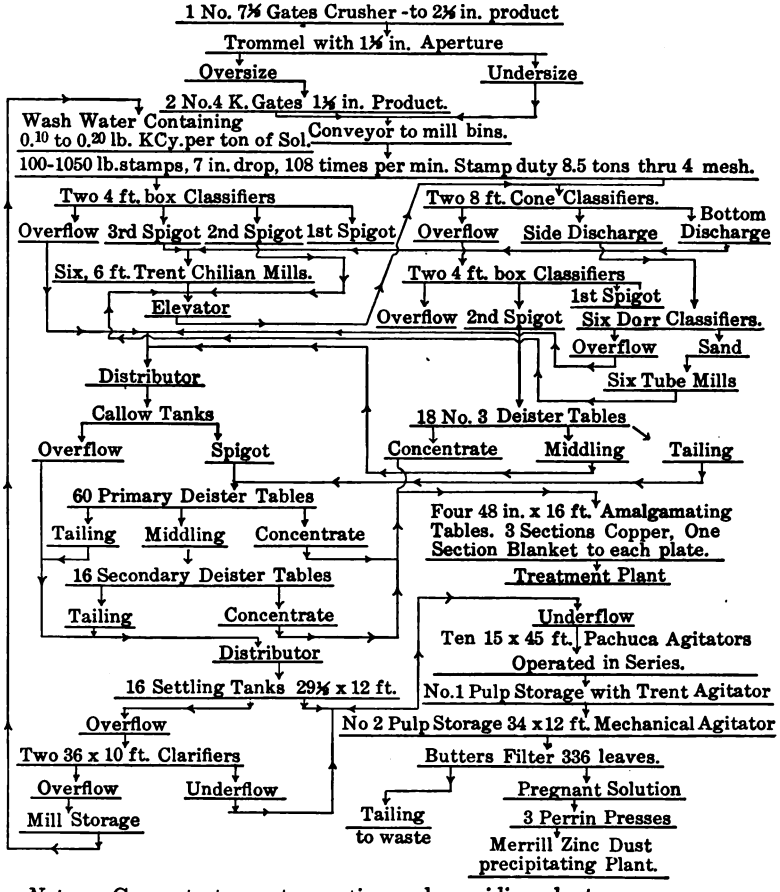
* Staff Correspondence, Min. & Sci. Press, Vol. 101, page 271.

(4) AMALGAMATION, CONCENTRATION, AND CYANIDATION

III. All Slime Cyanide Treatment. The mill of the Goldfield Consolidated Co.* at Goldfield, Nevada.

Character of Ore: — Siliceous dacite carrying complex sulphides of copper, lead and zinc in small amounts with occasional tellurides and always some free gold. Probably, on the whole, the most complex and difficult gold ore that is at present being handled by the cyanide process. Average value of ore steadily declining from $50 in 1910 to less than $20 in 1913.

Flow Sheet:



Note. — Concentrates go to roasting and cyaniding plant.

Results: — The total recovery is 94 per cent made up as follows: by amalgamation 17½ per cent, by concentration 54 per cent, and by cyanidation 22½ per cent. The total cost of operation and repairs for the entire treatment is $1.61. The consumption of chemicals is as follows: KCy 3.1 lb.; lime 8.5 lb.; zinc dust ½ lb.; and lead acetate ½ lb. per ton.

FIG. 111.

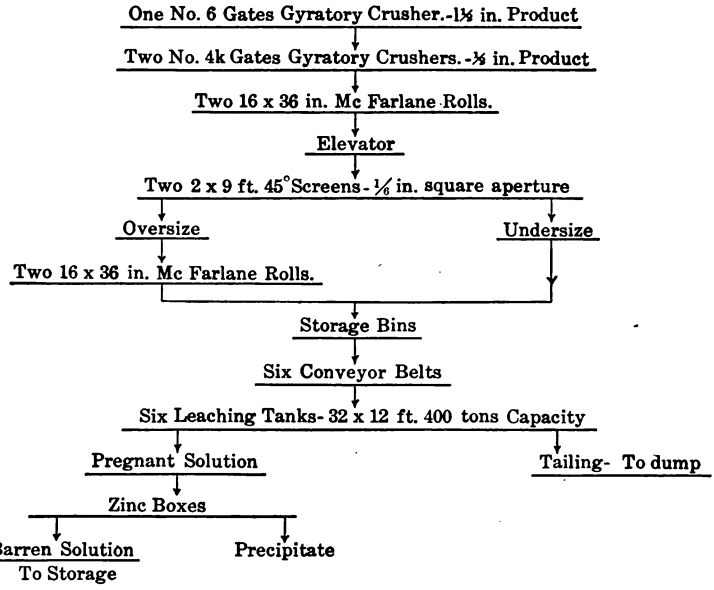
* J. W. Hutchinson, Min. & Sci. Press, Vol. 102, page 616, *et seq.*

(5) CYANIDATION

I. Leaching Unassisted by any other Process. It is rare to find an ore yielding a satisfactory portion of its gold and silver content by so simple a treatment. The mill of the Wasp No. 2 Mine, at Deadwood, S. Dakota,* represents the acme of simplicity in cyaniding.

Character of Ore: — The ore is a hard close grained Cambrian quartzite, the average value being $2.50 per ton, the ratio of silver to gold being roughly 2 to 1.

Flow Sheet:


One No. 6 Gates Gyratory Crusher.-1½ in. Product
Two No. 4k Gates Gyratory Crushers.-½ in. Product
Two 16 x 36 in. Mc Farlane Rolls.
Elevator
Two 2 x 9 ft. 45° Screens-⅛ in. square aperture
Oversize
Undersize
Two 16 x 36 in. Mc Farlane Rolls.
Storage Bins
Six Conveyor Belts
Six Leaching Tanks-32 x 12 ft. 400 tons Capacity
Pregnant Solution
Tailing- To dump
Zinc Boxes
Barren Solution
To Storage
Precipitate

Results: — The value of the ore is $2.50 per ton, the bullion recovered is $1.98 per ton, corresponding to 80 per cent extraction. The cost of treatment is 53 cts. per ton.

FIG. 112.

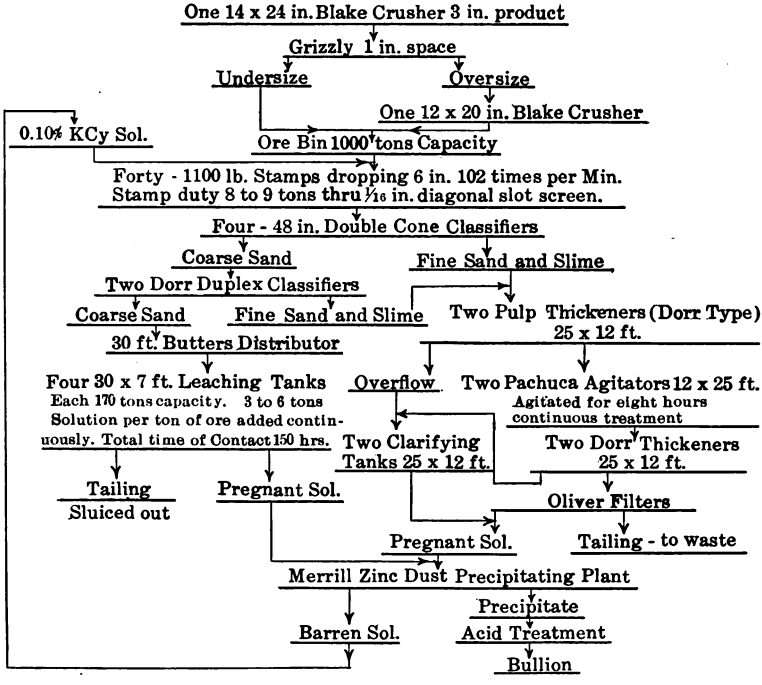
* Jesse Simmons, Eng. & Min. Jour., Vol. 92, page 214.

(5) CYANIDATION

II. Applied to the Treatment of both Sand and Slime. The plant of the Trinity Gold Mining and Reduction Co.,* at Carrville, Trinity Co., California, is a good example of this class.

Character of Ore: — "Oxidised soft and friable." Average value $5.00 to $5.50 per ton. Ore has changed materially since plant was built, which accounts for low extraction.

Flow Sheet:


One 14 x 24 in. Blake Crusher 3 in. product
Grizzly 1 in. space
Undersize
Oversize
One 12 x 20 in. Blake Crusher
0.10% KCy Sol.
Ore Bin 1000 tons Capacity
Forty - 1100 lb. Stamps dropping 6 in. 102 times per Min.
Stamp duty 8 to 9 tons thru 1/16 in. diagonal slot screen.
Four - 48 in. Double Cone Classifiers
Coarse Sand
Fine Sand and Slime
Two Dorr Duplex Classifiers
Coarse Sand
Fine Sand and Slime
Two Pulp Thickeners (Dorr Type) 25 x 12 ft.
30 ft. Butters Distributor
Four 30 x 7 ft. Leaching Tanks
Each 170 tons capacity. 3 to 6 tons Solution per ton of ore added continuously. Total time of Contact 150 hrs.
Overflow
Two Pachuca Agitators 12 x 25 ft.
Agitated for eight hours continuous treatment
Two Clarifying Tanks 25 x 12 ft.
Two Dorr Thickeners 25 x 12 ft.
Tailing
Sluiced out
Pregnant Sol.
Oliver Filters
Pregnant Sol.
Tailing - to waste
Merrill Zinc Dust Precipitating Plant
Precipitate
Barren Sol.
Acid Treatment
Bullion

Results: — The extraction is about 82 per cent. Cost of milling $1.01 (?).

FIG. 113.

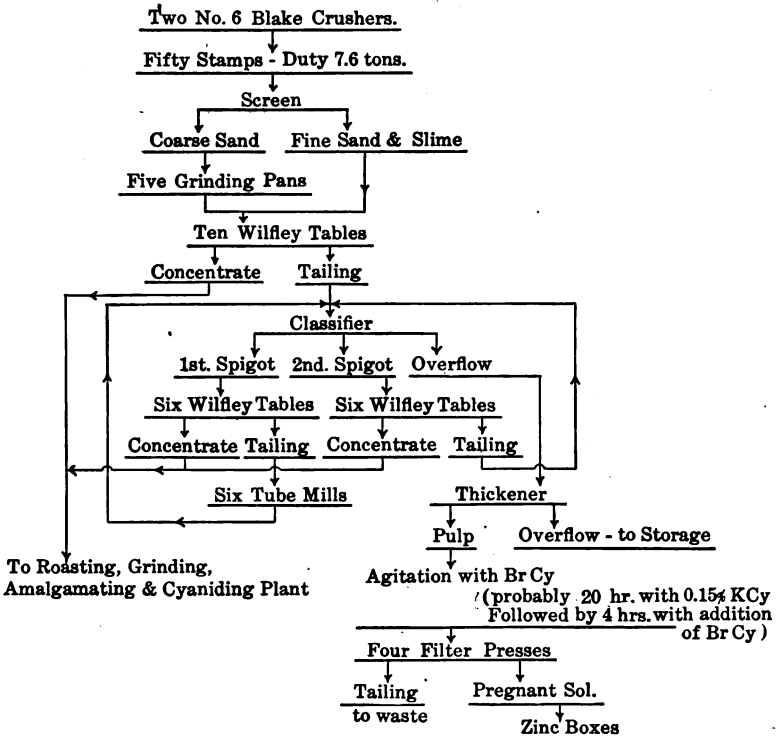
* J. W. Swaren, Eng. & Min. Jour., Vol. 91, page 907.

(7) CYANIDATION

V. With the Aid of an Oxidizing Reagent in the Solution. As discussed elsewhere, bromine is the only oxidizer which has so far found extensive and economic application. As in the preceding case we must look to West Australia for the best practice in this respect. The plant of the Oroya Brownhill Co.,* at Kalgoorlie, affords a typical example of what has been usually known as the Diehl process. (Fig. 116.)

Character of Ore: — Similar to preceding.

Flow Sheet:



Results: — The ore averages $9.35 per ton; the extraction is $8.70 or 93 per cent; the treatment cost is $2.50 per ton; 17 per cent of the recovery is made by concentration, the balance by bromo-cyanidation.

FIG. 116.

* A. E. Drucker, Min. & Sci. Press, Vol. 101, page 401.

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CHAPTER XV

TREATMENT OF SILVER ORES

The application of cyanide to silver metallurgy is a subject concerning which much less is known than of the corresponding treatment of gold ores. Rapid strides are being made, however, and the pioneer work done on the silver-ores of Mexico has re-

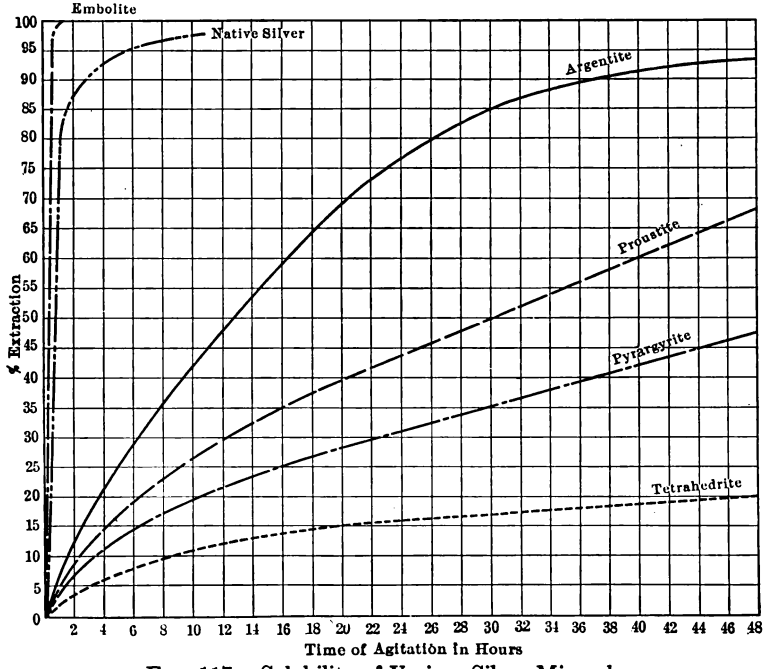


FIG. 117. Solubility of Various Silver Minerals.
(Conditions — 50 gm. ore, 100 cc. solution containing 0.5 per cent KCy and 0.3 per cent CaO.)

sulted in extensive silver cyaniding plants in that country and in the State of Nevada. As previously noted, silver occurs in a far greater variety of mineral forms and associations than does gold, and the solubility of the different silver minerals in cyanide solution also varies within wide limits. The preceding diagram (Fig. 117),

by Theo. P. Holt,* shows the relative solubilities of the six most important silver minerals: embolite, native silver, argentite, proustite, pyrargyrite, and tetrahedrite. It is to be noted at the outset that much stronger solutions of cyanide are required in the treatment of silver ores than are necessary in the cyaniding of gold ores.

Various oxidizers and extra solutions seem to promise better with silver than with gold. Fig. 118, also taken from Holt,† illustrates the effect of various reagents when added to cyanide

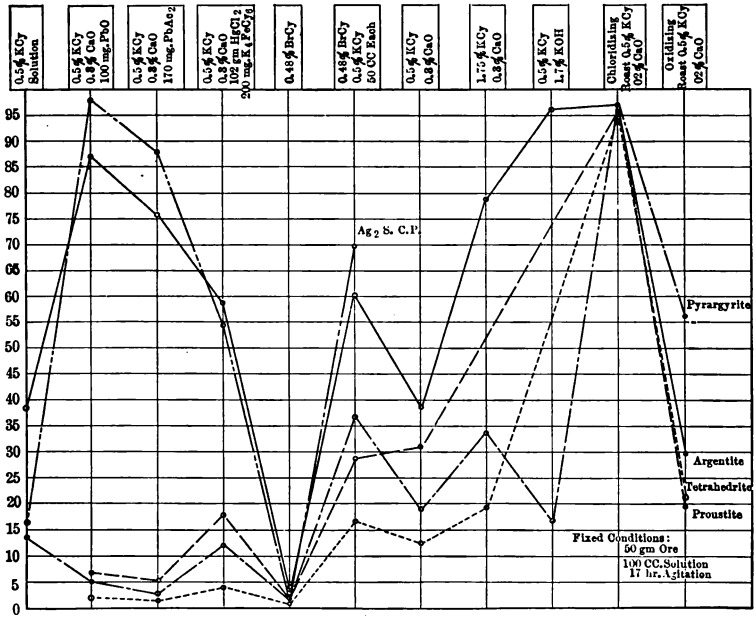


FIG. 118. Effect of Various Reagents in Cyaniding Silver Minerals.

solution in the treatment of artificially prepared mixtures of different silver-minerals and quartz sand. The main difficulty with all of the silver-sulphide minerals seems to be in the liability of the solution reaction, $Ag_2S + 4\ KCy \rightleftarrows 2\ KAgCy_2 + K_2S$, reversing itself, with the result that the silver is prematurely precipitated as sulphide. The effect of lead salts is to prevent this reversal by taking up the sulphur to form lead sulphide, thus getting

* Min. & Sci. Press, Vol. 99, page 159.

† Loc. cit.

it out of the way. It is to be noted that these results have not always been confirmed by other experimenters,* and that Holt's conclusions contradict those of Hobson † as to the beneficial effect of mercuric salts. The complex silver-ores of Cobalt are prepared for cyanide treatment by a preliminary desulphurizing reduction to the metallic state with hydrogen generated by caustic alkali and aluminium, in contact with the ore in a small tube-mill.‡

In crushing the ore and agitating it with solution, and in separating solution from pulp, the methods discussed herein for gold are, of course, readily applicable. The precipitation of silver from cyanide solutions is effected both by electrolysis and by zinc, the latter either in shaving or dust form. The bulk of the precipitate is, of course, many times greater than in treating a gold ore of corresponding value, and the consumption of zinc per ton of ore is also necessarily much greater. Aside from these points the precipitation of silver from cyanide solutions is, if anything, simpler than that of gold.

The discussion on page 227, concerning the liberation of the particle of gold-mineral, applies with equal force to the minerals of silver. In spite of the fact that the ores are more complex, however, the variation in methods of treatment, so far as cyaniding is concerned, are much fewer. Amalgamation is practically never used with cyanide; leaching, on account of the long time of contact, and large number of washes, is only rarely resorted to, and concentration with 'all-sliming,' agitation, and vacuum-filtering, bids fair to remain for some time the standard silver cyaniding practice.

Three types of plants will be shown in detail as representing the treatment methods in use:

(1) *Concentration, with leaching of sand and agitation of slime.*

(2) *Concentration, followed by cyanide treatment of an 'all-slime' product.*

(3) *'All-slime' cyanidation.*

* W. A. Caldecott, Min. & Sci. Press, Vol. 97, page 295.

† Francis I. Hobson, Min. & Sci. Press, Vol. 97, pages 159 and 182.

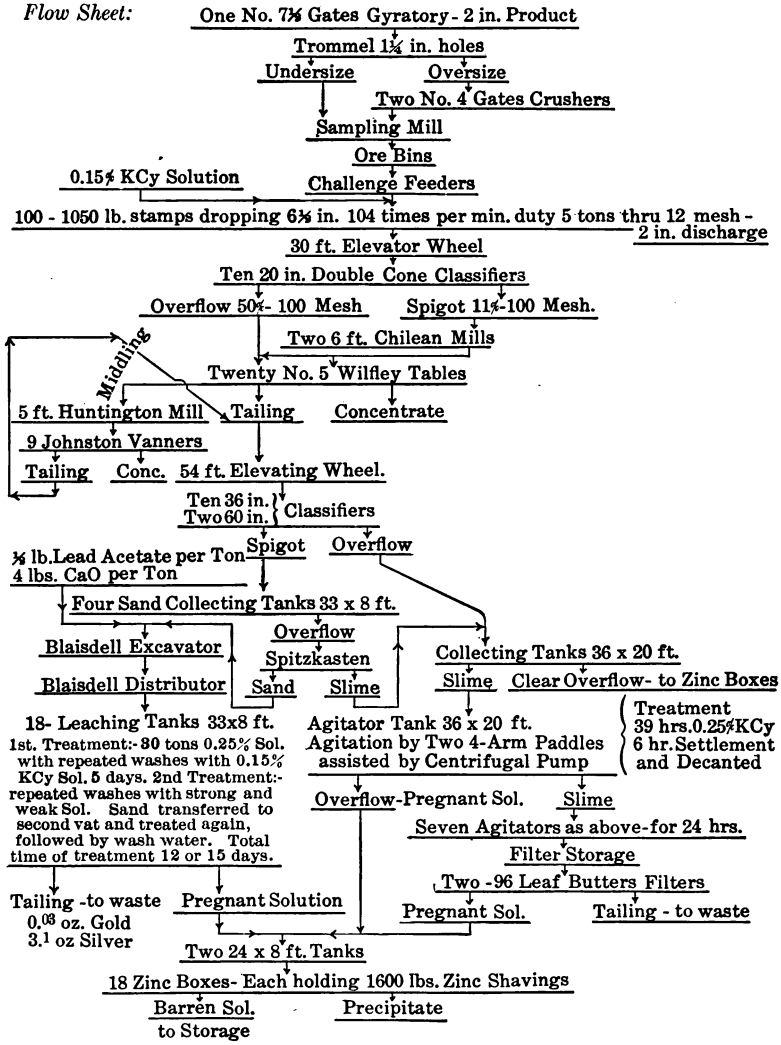
‡ Jas. J. Denny, Min. & Sci. Press, Vol. 107, p. 484.

(1) CONCENTRATION WITH LEACHING OF SAND AND AGITATION OF SLIME

The mill of the Tonopah Mining Co.,* at Millers, Nevada, commonly known as the Desert Mill, is a good example.

Character of Ore: — Siliceous andesite carrying gold and silver in the ratio of 1 to 100 by weight. The silver occurs mainly as stephanite, argentite, and polybasite with small amounts of galena, blende, pyrite, and chalcopyrite.

Flow Sheet:


One No. 7½ Gates Gyratory - 2 in. Product
Trommel 1¼ in. holes
Undersize
Oversize
Two No. 4 Gates Crushers
Sampling Mill
Ore Bins
0.15% KCy Solution
Challenge Feeders
100 - 1050 lb. stamps dropping 6½ in. 104 times per min. duty 5 tons thru 12 mesh - 2 in. discharge
30 ft. Elevator Wheel
Ten 20 in. Double Cone Classifiers
Overflow 50%- 100 Mesh
Spigot 11%-100 Mesh.
Two 6 ft. Chilean Mills
Middling
Twenty No. 5 Wilfley Tables
5 ft. Huntington Mill
Tailing
Concentrate
9 Johnston Vanners
Tailing
Conc.
54 ft. Elevating Wheel.
Ten 36 in.
Two 60 in.
Classifiers
Spigot
Overflow
½ lb. Lead Acetate per Ton
4 lbs. CaO per Ton
Four Sand Collecting Tanks 33 x 8 ft.
Overflow
Blaisdell Excavator
Spitzkasten
Blaisdell Distributor
Sand
Slime
Collecting Tanks 36 x 20 ft.
Slime
Clear Overflow - to Zinc Boxes
18- Leaching Tanks 33x8 ft.
1st. Treatment:- 80 tons 0.25% Sol. with repeated washes with 0.15% KCy Sol. 5 days. 2nd Treatment:- repeated washes with strong and weak Sol. Sand transferred to second vat and treated again, followed by wash water. Total time of treatment 12 or 15 days.
Agitator Tank 36 x 20 ft.
Agitation by Two 4-Arm Paddles assisted by Centrifugal Pump
Treatment 39 hrs. 0.25% KCy 6 hr. Settlement and Decanted
Overflow-Pregnant Sol.
Slime
Seven Agitators as above - for 24 hrs.
Filter Storage
Two - 96 Leaf Butters Filters
Tailing - to waste 0.03 oz. Gold 3.1 oz Silver
Pregnant Solution
Pregnant Sol.
Tailing - to waste
Two 24 x 8 ft. Tanks
18 Zinc Boxes - Each holding 1600 lbs. Zinc Shavings
Barren Sol. to Storage
Precipitate

Results: — 1907 ore assayed $52. Extraction 92%. Cost $2.94 per ton.
1912 ore assayed $19.97. Extraction 90.6%. Cost $2.74 per ton.

FIG. 119.

* A. R. Parson, Min. & Sci. Press, Vol. 95, pages 495–500.

(2) CONCENTRATION FOLLOWED BY CYANIDE TREATMENT OF AN 'ALL-SLIME' PRODUCT

The mill of the Montana-Tonopah Co., at Tonopah,* Nevada, is an excellent example; since, however, one Tonopah plant has already been discussed, the mill of the San Rafael Co., at Pachuca, Hidalgo, Mexico,† will be outlined.

Character of Ore: — The ore is composed of pyrite, blende, and galena, in a gangue of quartz and calcite, the silver occurring mainly as argentite. The ore assays from 4 to 5 grams gold and 900 grams of silver to the metric ton.‡

Flow Sheet:

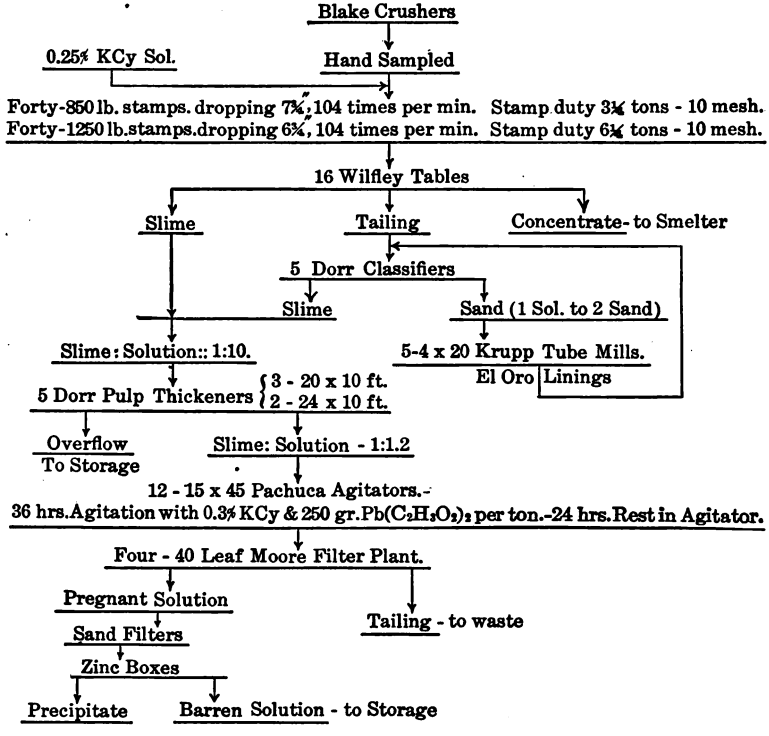

Blake Crushers
0.25% KCy Sol.
Hand Sampled
Forty-850 lb. stamps. dropping 7¾", 104 times per min. Stamp duty 3¼ tons - 10 mesh.
Forty-1250 lb. stamps. dropping 6¾", 104 times per min. Stamp duty 6¼ tons - 10 mesh.
16 Wilfley Tables
Slime
Tailing
Concentrate - to Smelter
5 Dorr Classifiers
Slime
Sand (1 Sol. to 2 Sand)
Slime: Solution:: 1:10.
5-4 x 20 Krupp Tube Mills.
El Oro Linings
5 Dorr Pulp Thickeners
3 - 20 x 10 ft.
2 - 24 x 10 ft.
Overflow
To Storage
Slime: Solution - 1:1.2
12 - 15 x 45 Pachuca Agitators.
36 hrs. Agitation with 0.3% KCy & 250 gr. Pb(C2H3O2)2 per ton. - 24 hrs. Rest in Agitator.
Four - 40 Leaf Moore Filter Plant.
Pregnant Solution
Sand Filters
Zinc Boxes
Tailing - to waste
Precipitate
Barren Solution - to Storage

FIG. 120.

* C. H. Rotherham, Min. & Sci. Press, Vol. 97, page 324.

† E. Girault, Min. and Min., Vol. 30, page 618.

‡ Grams per metric ton is the usual method of expressing assay returns in Spanish America. 1 gm. = 0.0001 per cent of a metric ton, or is equivalent to 0.2916603 oz. Troy per Avoirdupois Ton. In this ore the gold content is between 0.117 and 0.146 oz., the silver is 26.2 oz.

Results: — The extraction is 97% of the gold and 91.5% of the silver; 22% is made by concentration, the balance by cyanide. The cyanide consumption is 1.078 kilo of NaCy (128 per cent KCy) per metric ton.

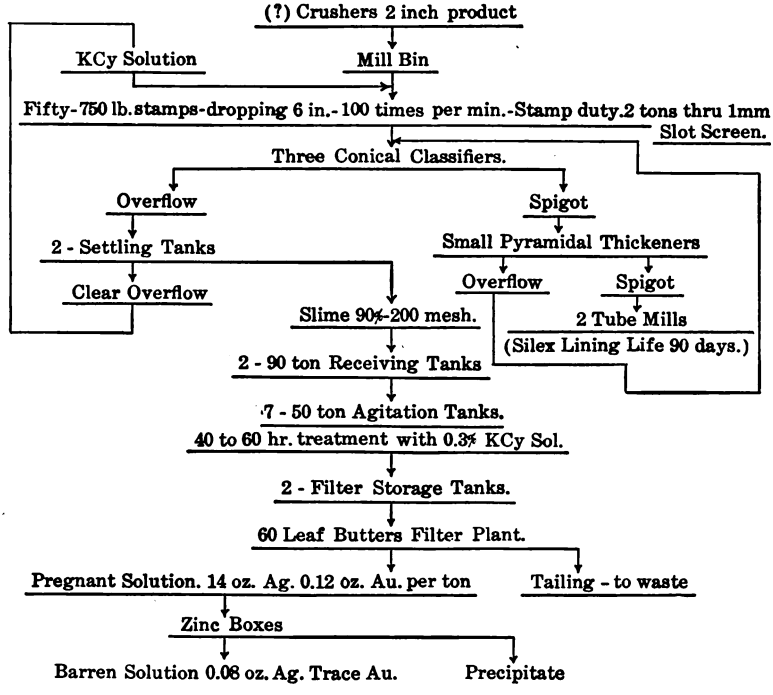
The cost is reported as follows:

General expense	$0.39
Grinding and Concentrating	0.59
Cyaniding	0.66
Pumps and compressor	0.075
Filtration (with royalty)	0.165
Precipitation and melting	0.22
Total	$2.100

(3) 'ALL-SLIME' CYANIDATION

All-slime cyanide treatment, without the assistance of concentration, is only possible, at present, with the simpler silver ores. As knowledge increases, it may be possible to extend this method to the more complex ores. The mill of the New York and Honduras Rosario Mining Co.,* at San Juancito, Honduras, recently abandoned the amalgamation process in favor of cyanidation, and their scheme of treatment is shown below.

Character of Ore: — Hard quartz carrying "lead, copper, iron, and antimony."
Flow Sheet:


(?) Crushers 2 inch product
KCy Solution
Mill Bin
Fifty-750 lb. stamps-dropping 6 in.-100 times per min.-Stamp duty.2 tons thru 1mm Slot Screen.
Three Conical Classifiers.
Overflow
Spigot
2 - Settling Tanks
Small Pyramidal Thickeners
Clear Overflow
Overflow
Spigot
Slime 90%-200 mesh.
2 Tube Mills
(Silex Lining Life 90 days.)
2 - 90 ton Receiving Tanks
7 - 50 ton Agitation Tanks.
40 to 60 hr. treatment with 0.3% KCy Sol.
2 - Filter Storage Tanks.
60 Leaf Butters Filter Plant.
Pregnant Solution. 14 oz. Ag. 0.12 oz. Au. per ton
Tailing - to waste
Zinc Boxes
Barren Solution 0.08 oz. Ag. Trace Au.
Precipitate

Results: — The ore assays $24. Ratio Ag to Au, 10 to 1 by weight. The total extraction is 90 per cent. The cost of treatment is $3.46 per ton. The cyanide consumption is extremely high, — nearly 8 lb. per ton.

FIG. 121.

* Geo. E. Driscoll, Min. & Sci. Press, Vol. 98, page 388.

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